COAL DIVISION

1942

A. I. M. E.

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Columbus, O.

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FOREWORD

This eighth Coal Division volume includes many of the papers and discussions from the Division's regular fall and winter sessions, through the administrations respectively of Chairman Charles E. Lawall in 1940 and of Chairman J. E. Tobey in 1941. The inclusive period opened with the fall meeting at Birmingham in 1940 and closed with the annual New York meeting of 1942. The joint fall meetings with the Fuels Division of the American Society of Mechanical Engineers, at Birmingham in 1940 and at Easton in 1941, continued the progression of value and attraction that have successively marked these joint programs since their beginning.

The Division's executive committee has shown sharp awareness of the membership's best interest and has stoutly supported the officers in efforts to make the Division's activities of most value to the members and to the Coal Industry.

For their faithfulness, we are grateful to Secretary David R. Mitchell and to Programs Chairman Henry F. Hebley; they have willingly borne the heat and burden of the day for the Division.

Programs are being planned for the greatest practical contribution to the war effort.

Both the Division and the Institute are indebted to the technical committees for continuing to arouse interest in the newer phases of progress in coal as well as to the authors of the following papers.

NEWELL G. ALFORD,
Chairman, Coal Division.

PITTSBURGH, PA.
September 1, 1942.

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Development With and Against the Pitch at Coal Mines in Southwestern Wyoming

By John E. Willson* and Frank P. Lebar†

(New York Meeting, February 1941)

Typical of southwestern Wyoming are coal structures that dip from 4° to 17°. Those at the Reliance and Winton mines of the Union Pacific Coal Co. average 9½° and 15°, respectively, and dip almost directly to the west. By using a hoist for main haulage, a method of mining has been developed whereby four places are driven on the pitch of the seam. Slightly upgrade from the strike at intervals of 300 ft. a two-entry system is driven to the north and to the south. In both fields, which are nongassy, an exhaust system of ventilation is used. Slope, manway, and lower or haulage entries are placed on the intake. Two aircourses and upper or breaking entries are placed on the return. A retreat system of mining is used exclusively.

From the opening of its first coal mine, in the year 1868, until 1923, when the Joy loader was introduced into The Union Pacific Coal Company's operations, slopes, aircourses, and manways were driven by hand. Development work was conducted on the night shift or off shift, and the main hoist was used for haulage. The slope was driven down the pitch with crosscuts turned at intervals of 75 ft. and entries at 230 ft. Connections to the workings above were made by driving up the pitch 75 ft. Mule haulage was used to furnish cars to the hand loaders driving planes up the pitch. A sheave was placed at the upper end of the plane with a 1¼-in. hemp rope doubling around it and having one end attached to the car and the other to the mule. The mule, pulling downhill, raised to the working face a small wooden car of one-ton capacity.

With the introduction of mechanical loading, mule haulage and small wooden cars disappeared and auxiliary hoists, electric locomotives, and steel cars with a capacity of 4 tons were substituted. Development work was done during two or three shifts of the 24-hr. period. Driving of haulage slopes down the pitch by hand continued. Crosscut intervals were changed to 100 ft. and the crosscuts were driven with shaking conveyors. With the use of a 90° swivel, aircourses and manways were driven up the pitch. Entry intervals were changed from 230 ft. to 300 feet.

Mobile loaders of the Joy type were first employed by this company in the year 1923 at Hanna, Wyo. In 1928, two 4-BU Joy loaders were transferred from Hanna to Winton No. 1 mine, where they were used in driving slopes and entries. At the time of transfer, the machines were about worn out and were not suited to the steep pitch, therefore frequent breakdowns occurred. However, experience justified the belief that the mobile loader was fitted for driving slopes in this field. During 1937 a unit was installed in Reliance No. 7 mine, and during 1939 one in Winton No. 7½ mine. With the introduction of this type of mobile loader, a new era of development began. Speed undreamed of in hand-loading days became a reality. The cost of development down the pitch has been reduced until it

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compares favorably with that of driving up the pitch with shaking conveyors. A comparison of costs is as follows:

	Per Cent
Hand loading down the pitch, as basis....	100.00
Hand loading up the pitch................	92.36
Driving down the pitch with mobile loaders	64.24
Driving up the pitch with shaking conveyors..............................	59.66

All calculations are based on the present scale of wages and on cost per ton.

year, whereas with the new method 1000 ft. are driven in three to four months. Enough slope can be extended in a summer or slack period to permit five years of mining. During busy periods the mobile loader is used in room work as the major producing unit.

To service the mobile loader with cars, an auxiliary hoist is set in the crosscut between the north aircourse and the slope,

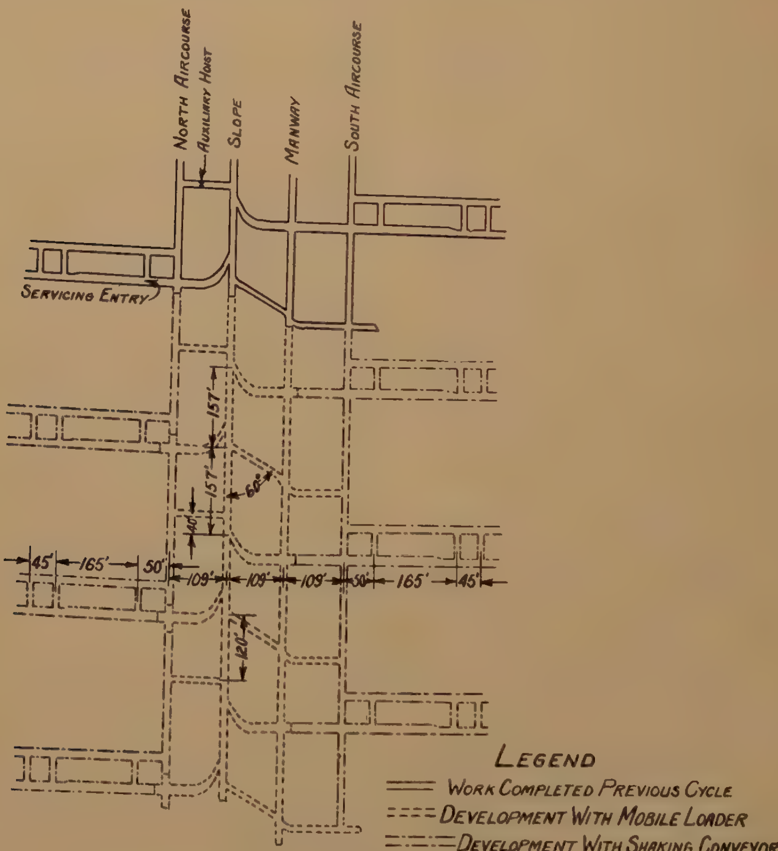

Fig. 1.—Development plan.

Two places, slope and manway, instead of one, are now driven down the pitch, thus permitting a more positive system of ventilation and eliminating the waiting period of driving up the pitch. The new method thus expedites slope driving. Using the old method, 600 ft. were driven in a

just above the lowest north entry. It remains in this crosscut during the complete stage of development. Six empty cars are lowered by the main hoist onto the parting of the entry (Fig. 1). Three are picked up by the auxiliary hoist and dropped to the mobile loader. When the first car is filled,

the trip is pulled up to the servicing entry, where the load is left on the lower track. Two empties are then lowered to the unit and the procedure is repeated until the three-car trip is loaded. A new trip of three empties is loaded in the same manner. When the six cars have been loaded, the main rope lowers six new empties into the parting and removes the loaded trip. Contact is maintained with the main hoist to avoid delay in switching. As development work is farther removed from the auxiliary hoist it becomes less feasible to take each individual car to the parting, therefore cars are switched onto a side track near the mobile loader until a three-car trip is loaded.

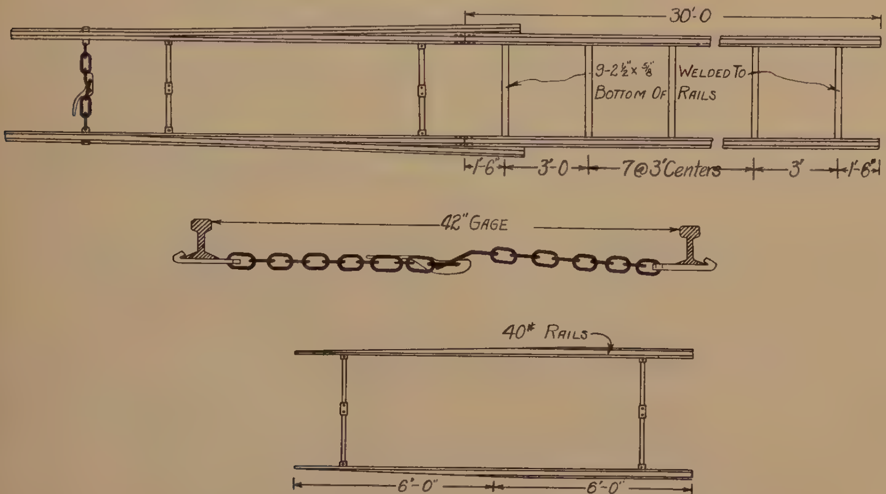

FIG. 2.—TRACK EXTENSION FOR MOBILE LOADER.

The previous cycle of development was ended by leaving a 60° slant to the south of the slope and connected to the manway (shown in Fig. 1). In the first stage of development the manway is driven 20 ft. below the slant. From this point a crosscut is driven 109 ft. to the south on level grade, and when the crosscut is completed the manway is extended an additional 280 ft. The second stage consists of driving the slope 157 ft. below the slant, at which point a south entry is turned and driven to intersect the manway. Upon the completion of the entry, the slope is driven an additional 157 ft. and a new slant is turned. This process is repeated until three slants and entries have been turned to the south. In the third stage, north entries and crosscuts are driven. Working from the bottom of the slope upward, entries are turned at 40 ft. above slants while crosscuts are turned at 40 ft. above south entries. North crosscuts are driven 109 ft., and north entries far enough beyond the aircourse for three cars.

In developing, the loading and preparation processes are not alternated at two places. By continuous loading and cutting in the same place, the use of a pump is eliminated. Pumping is necessary only when a place stands idle for a period of time.

Rails are kept to the face by means of a 30-ft. sliding extension (Fig. 2) used in conjunction with 40-lb. temporary track. Temporary track is advanced during the cutting or preparation process. As development proceeds, temporary rails are replaced with 75-lb. steel and turnouts are installed as the permanent track is laid.

The mobile loader is a 7-BU Joy, and cutting is done with a Goodman 12-AB mining machine. Cuttings are removed from the kerf with long-handled shovels.

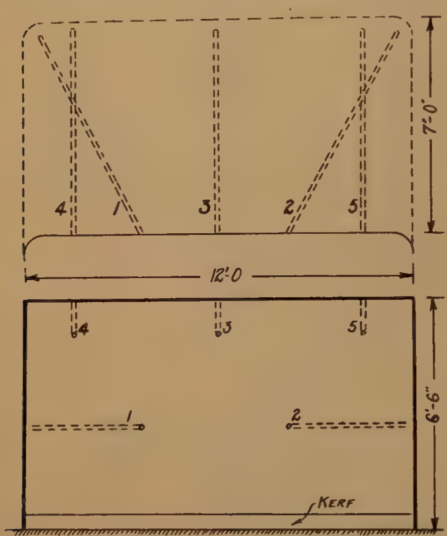

FIG. 3.—DRILLING PLAN. NUMBERS DENOTE ORDER OF SHOOTING.

Water is used on the cutter bar wherever a sufficient supply to wet the cuttings is not found. The mining machine is moved rapidly from place to place by mounting

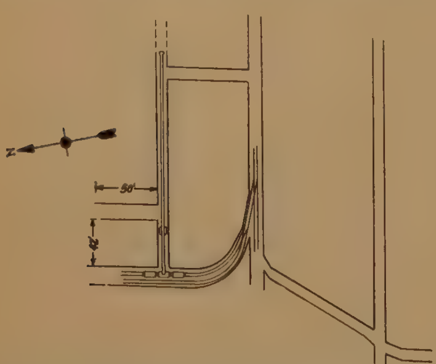

FIG. 4.—SHAKING CONVEYOR DRIVE FROM UPPER NORTH ENTRY.

it on a truck. Five holes (Fig. 3) are drilled at the face, using a Chicago "Little Giant" drill and McLaughlin auger with removable bits. Holes are tamped to the collar with dummies of incombustible material. One hole is shot at a time, using permissible powder. Ventilation beyond the last crosscut is maintained with an auxiliary fan of

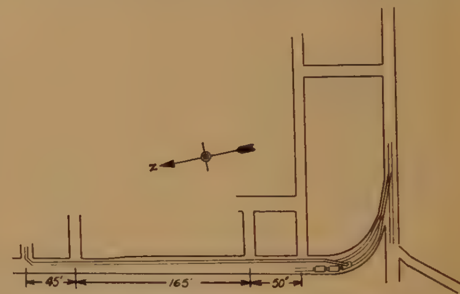

FIG. 5.—SHAKING CONVEYOR DRIVE FROM HAULAGE ENTRY.

the Jeffrey 2-hp. type, and the tubing is kept within 25 ft. of the face at all times. During the period of development, the haulage slope is used as an intake aircourse and the manway as a return. The crew consists of a Joy operator, a machineman, two track layers, a hoistman, a rope rider, and a unit foreman.

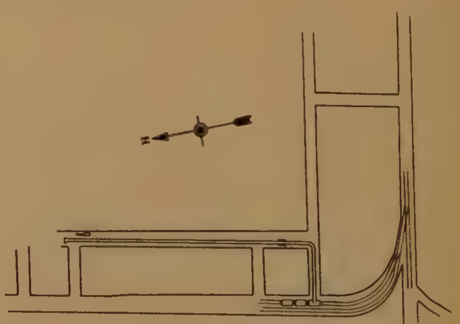

FIG. 6.—SHAKING CONVEYOR DRIVE FROM BREAKING ENTRY.

In Winton No. 7½ mine, the seam is about 7½ ft. thick; 6 ft. are mined and 1½ ft. left for roof protection. Reliance No. 7 mine is about 8½ ft. high. At about 6 ft. 8 in. from the bottom a ½-in. yellow band is found. Coal is mined to this yellow band, the rest being left as top coal. Slope and manway are driven 12 ft. wide and crosscuts 10 ft. wide. Entry width is 8 ft. where it leaves the slope and is gradually increased

to 17 ft. for the maintenance of clearance and the installation of double track on the parting. The entries are narrowed to 14 ft. beyond the double track.

After the mobile loader is removed from development work and placed in rooms, a shaking conveyor is moved into the upper north entry of the cycle. It drives up the pitch to connect with the crosscut and entry above (Fig. 4). When the face of the aircourse is up far enough to maintain a pillar of 42 ft., the breaking entry is turned and driven 50 ft., enough for a setup. The shaking conveyor is next set up in the haulage entry (Fig. 5), which is driven 260 ft. beyond the aircourse. Three crosscuts are driven up the pitch as the entry advances; the first at a point 50 ft. beyond the aircourse, so as to connect with the face of the breaking entry; the second, at 215 ft. and the third at 260 ft. The shaking conveyor is then moved to the breaking entry (Fig. 6) between the aircourse and the first crosscut, and loads down the aircourse into cars on the haulage entry. The pair of entries is then ready for future development. Dust barriers and overcasts can be installed while the entry is not in use. In a like manner the remainder of the north side is driven. The shaking conveyor is set up in the upper south entry between the slope and the manway. The entry is driven 45 ft. beyond the south aircourse, which is necked when driving in. A crosscut is driven up the pitch 42 ft. at the face of the entry (Fig. 7). From this point the same procedure is followed in developing as was followed on the north side. When the two lower south entries are completed, the shaking conveyor is set in the aircourse at the lower crosscut and the aircourse is driven to connect with the entry above. A Goodman G-15, 15-hp. shaking conveyor and a Goodman 112-CA mining machine are used for this work.

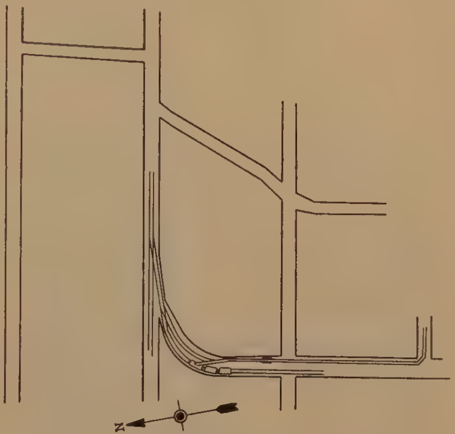

FIG. 7.—DEVELOPMENT UP THE PITCH, 42 FT. AT FACE OF ENTRY.

Drilling and ventilation are accomplished in the same manner as in driving down the pitch. Cars are furnished to the shaking conveyor by a method similar to that of servicing the mobile loader and three cars are loaded before hoisting to the servicing entry.

With the cycle completed, six entries have been turned and made ready for future development. Coal has been blocked out for a period of five years of mining, and the complete process has taken from six to nine months. A reserve is on hand, so that the mine may be expanded rapidly in case of unusual demand, or the portion developed may be left standing until needed in normal use.

Mining Anthracite on Pitching and Flat Seams over Mined-out Areas

By W. H. Moore* and E. T. Powell,† Members A.I.M.E.

(New York Meeting, February 1941)

In the early days of mining in the Anthracite field, only the thicker and better seams of coal were mined, because of the limited mining and coal-cleaning facilities, therefore many of the thinner and less productive seams were permitted to remain unmined. With improvements in mining methods, the introduction of modern preparation plants, and the increased market demand for small-size coal, the thinner and more laminated seams have now become assets. In many instances these seams overlie the mined-out beds of the thicker and better seams, from which all recoverable coal was removed by the conventional breast-and-pillar method of mining. Some collieries in certain sections of the Anthracite field are now relying on these thinner and laminated seams.

For the purpose of illustration, a large area in the Hickory Swamp-Ridge Basin of the western middle coal field has been selected, where as early as 1898 the No. 4, or Little Buck Mountain, seam was mined and pillars removed. The No. 5, or Buck Mountain, seam lying on a comparatively light pitch and with intervening strata of but 55 ft. of rock was not mined until 1935. Farther east, in the same basin the Nos. 8 and 9 seams, which are two splits of the Mammoth seam, have been mined and robbed and the workings have been abandoned since 1896. The overlying No. 9½ seam, or the upper split of the Mammoth, and the No. 9¾ or 4-ft. seam remained virgin, with only an interval of 50 ft. of rock separating them, and with 40 to 50 ft. of rock strata separating the No. 9½ seam from the completely mined area of the Nos. 8 and 9 seams. These seams lie on a pitch of 45° on the south side of the basin and on 75° on the north side of the basin.

Mining has been carried on in the 9½ and 9¾ seams for 3 years on both dips of the basin, for a distance of 3500 ft. In some sections the overlying strata of the thicker seams previously mined have settled without any apparent breaks in the upper strata, as indications have shown in the top split of the Mammoth; while in other areas the roof and bottom of this top split of the Mammoth have settled unevenly, thereby causing the bottom and roof to break into blocks.

In attempting to mine over a large area of the No. 5 or Buck Mountain seam, a similar settlement has been encountered; the roof of the seam has been broken and a considerable amount of timbering has been necessary in order to make the mining of the seam possible. The conventional breast-and-pillar method, using shaker conveyors, has been used in the No. 5 seam, and a slant-breast method in the mining of Nos. 9½ and 9¾ seams.

Mining of Light-pitch Seams

The No. 4 seam was mined and robbed by the conventional breast-and-pillar system. In this area the seam pitched from 5° to 25° on the north dip, averaged 8 ft. in thickness and had a slate bottom and a

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† Susque Colly Co., Glen Lyon Colliery, Glen Lyon, Pa.

sandstone roof, as shown in Fig. 1. Because of the strong top, a high percentage of coal was recovered from this area. The few pillars that were left intact offered some support to the overlying strata. The settling of these strata was the real cause of the breaking of the roof of the overlying seam.

No thought was given at that time to the mining of the overlying No. 5 seam, which in the earlier days was considered to have too many slate partings to make it commercially recoverable. However, in 1935 it was decided to mine the No. 5 seam from a rock slope previously driven, which divided in the center the area to be mined. West of this rock slope the seam dipped to the north from 5° to 15°; while east of the slope the seam dipped from 15° to 25°, as shown in Fig. 2. Gangways were driven east and west from the bottom of this slope, as shown in Fig. 3. On the east side, a plane designated on Fig. 3 as No. 5 seam plane was driven 1800 ft. up the pitch, and from this plane gangways were driven east and west at intervals so that breasts would be approximately 400 ft. long.

Chutes were driven off these gangways on 50-ft. centers at right angles to the strike of the seam. These chutes, driven 10 ft. in width, were connected by a monkey heading, through which the ventilating current was carried. Breasts or chambers were driven from the monkey heading starting at the width of the chute and increasing 24 ft. in width wherever roof conditions would permit. A shaker conveyor was installed in the center of the breast and the coal as it was blasted out was loaded on the conveyor, which carried it to the gangway loading it into mine cars. A brattice was maintained on the inside of the breast from the last heading to the face, which conducted the air current to the face of the breast, from which the air current crossed the breast and passed through the topmost heading to the adjoining breast. The cross headings were driven on the standard distance of 60 ft. as the breast advanced.

Because of the robbing or removal of pillars in the No. 4 seam underneath this

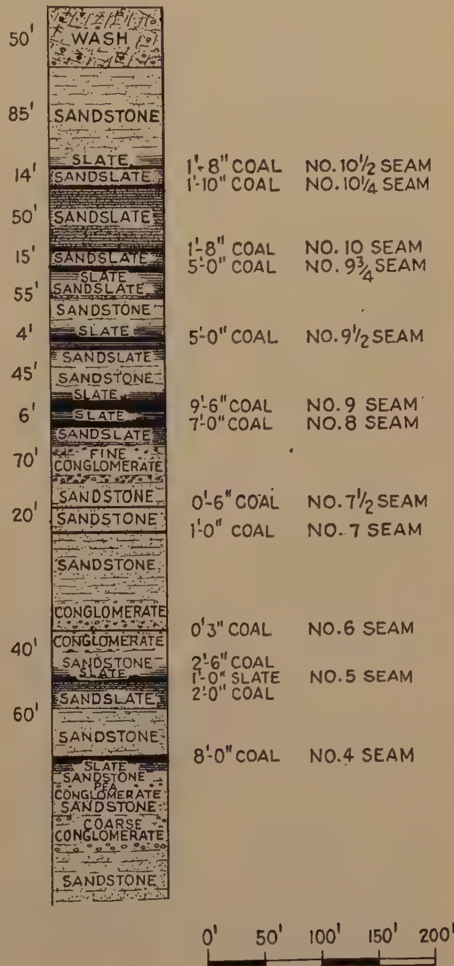

FIG. 1.—AVERAGE COLUMNAR SECTION OF STRATA THROUGH SWAMP-RIDGE BASIN.

area, four rows of props were placed in each breast, and in many places a large number of additional props were required to support the broken roof. At other places where mining was carried on in this seam and where the No. 4 seam had not been robbed, two rows of props were sufficient to support the roof.

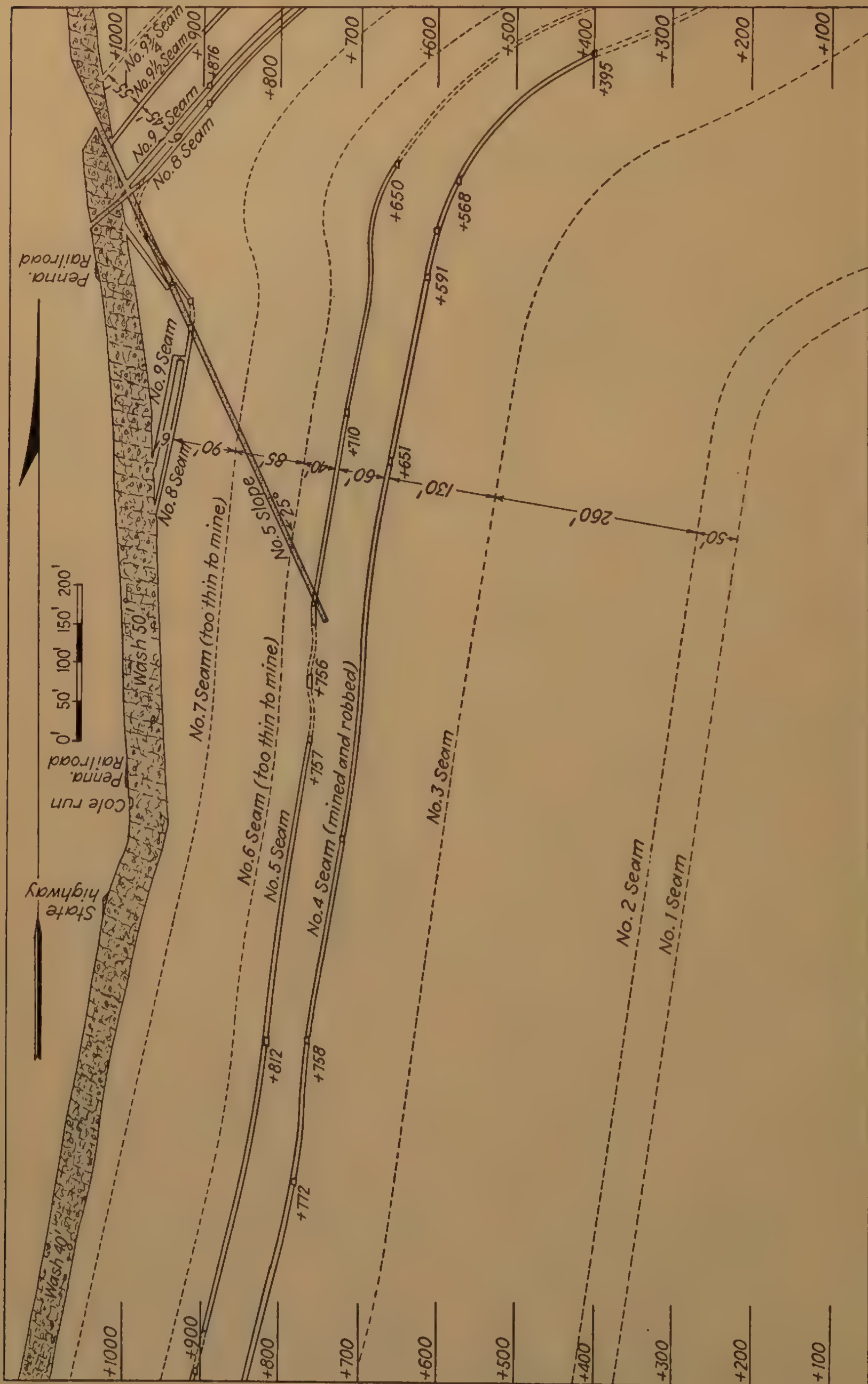

FIG. 2.—CROSS SECTION *A-A'* THROUGH FIG. 3. TYPICAL CROSS SECTION THROUGH AREA.

FIG. 3.—WORKINGS IN NO. 5 SEAM, HICKORY SWAMP SECTION, SHOWING METHOD OF MINING AND ROBBING.

This seam has two main benches of coal. The upper bench is 2 ft. 6 in. thick and the lower bench 2 ft. thick, with a middle slate varying from 8 to 20 in. in thickness, which is characteristic of this seam in the western middle field. Both benches have several slate partings which vary from ½ to 2 in. in thickness. The upper bench is blasted out first and loaded on the conveyor. The slate is then blasted out, and gobbed on both sides of the conveyor to prevent mixing with the coal, thus saving the loading and handling of the refuse in the mine car and on the surface. The lower bench is then blasted out and loaded on the conveyor. As the breasts advanced, the bottom and roof conditions encountered required continual inspection; in many places it was found that the seam was drawn away from the roof from 3 to 8 in.; and in other instances, the bottom slate had drawn away from the coal. Observations of the immediate roof overlying the No. 5 seam showed that the robbing of the No. 4 seam had caused a sag or bending of the immediate rock cover over the No. 5 seam, which was the main difficulty to be overcome in the mining of this seam. It was apparent that the large rock mass overlying the immediate roof had withstood the settlement or sagging, as no general caving or squeezing was noted throughout an area of approximately 4000 by 2000 ft. The pillars were removed by starting at the face of the breast. As a pillar was removed and larger areas of roof exposed, a large amount of timber was required to prevent the immediate roof from caving to near the working face. As the robbing continued, the roof fell in large masses close to the safety props.

Even considering the difficulties arising from previous mining of the under seam, it has been estimated that the ultimate extraction was about 90 per cent of the original coal in place. However, this recovery was obtained at a much greater cost than would have been necessary if the No. 4 seam had not been robbed.

Slant Breast-and-pillar Method

To the east in the same basin the following conditions are found; the average pitch of the seams on the north dip is 48° and on the south dip 75°, as shown on Fig. 4. The average thickness of coal in Nos. 9½ and 9¾ seams is 4 ft. 6 in. and 5 ft. 2 in., respectively. No. 9½ seam has a slate or clod roof 1 ft. 10 in. thick, above which is the main roof of slate. No. 9¾ seam also has a clod roof, which is 2 ft. 8 in. thick to the main roof of sandslate.

Underlying the seams mentioned were Nos. 8 and 9 seams, or Mammoth bed, which was mined and robbed by the breast-and-pillar method. No. 9 seam, as shown in Fig. 5, was mined and robbed through rock holes driven from the No. 8 seam gangway on the north dip and off gangways driven in the seam on the south dip. After No. 9 seam was completely mined and robbed, No. 8 seam was robbed. Nos. 8 and 9 seams were 9 ft. 6 in. and 7 ft., respectively, with an intervening rock stratum of 6 ft.

Gangways on both dips in Nos. 9½ and 9¾ seams were timbered with three-piece sets, spaced 6½ ft. between collar notches, 7 ft. clear above the rail to the underside of the collar and 11 ft. in the clear between the legs at the rail. These sets were spaced on 5½-ft. centers. This method of gangway timbering was made necessary by the draw from the underlying workings. In areas where the underlying seam was not removed first, gangways were timbered by using only props or a post and bar.

When the conventional method of breast-and-pillar mining was introduced in Nos. 9½ and 9¾ seams (Fig. 6), it was found that as the coal was removed the clod that had previously been disturbed by the extraction of Nos. 8 and 9 seams would break and mix with the coal. To prevent this it was proposed to work the breasts

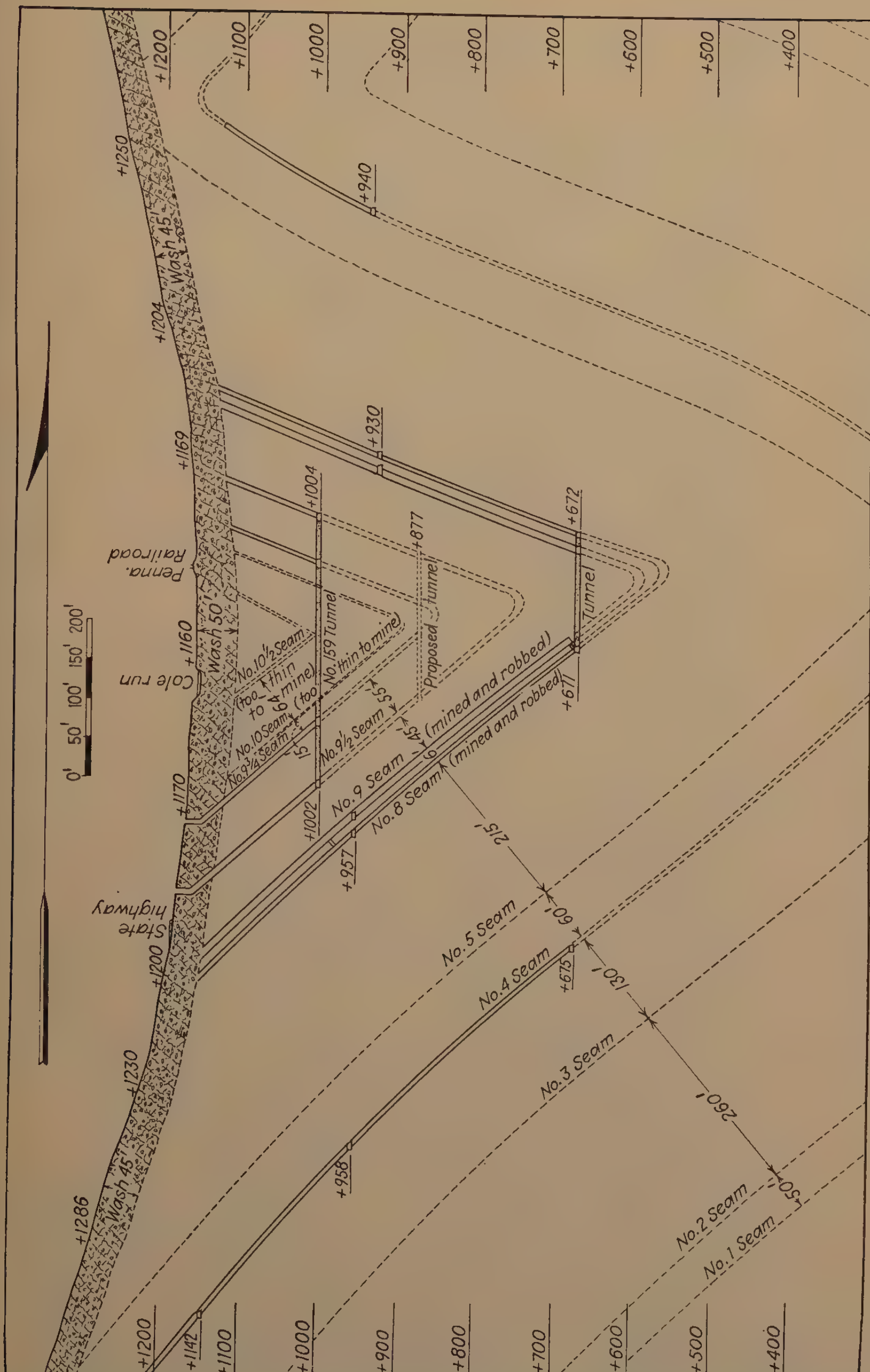

FIG. 4.—CROSS SECTION *B-B'* THROUGH FIGS. 5 AND 6. TYPICAL OF THIS LOCATION.

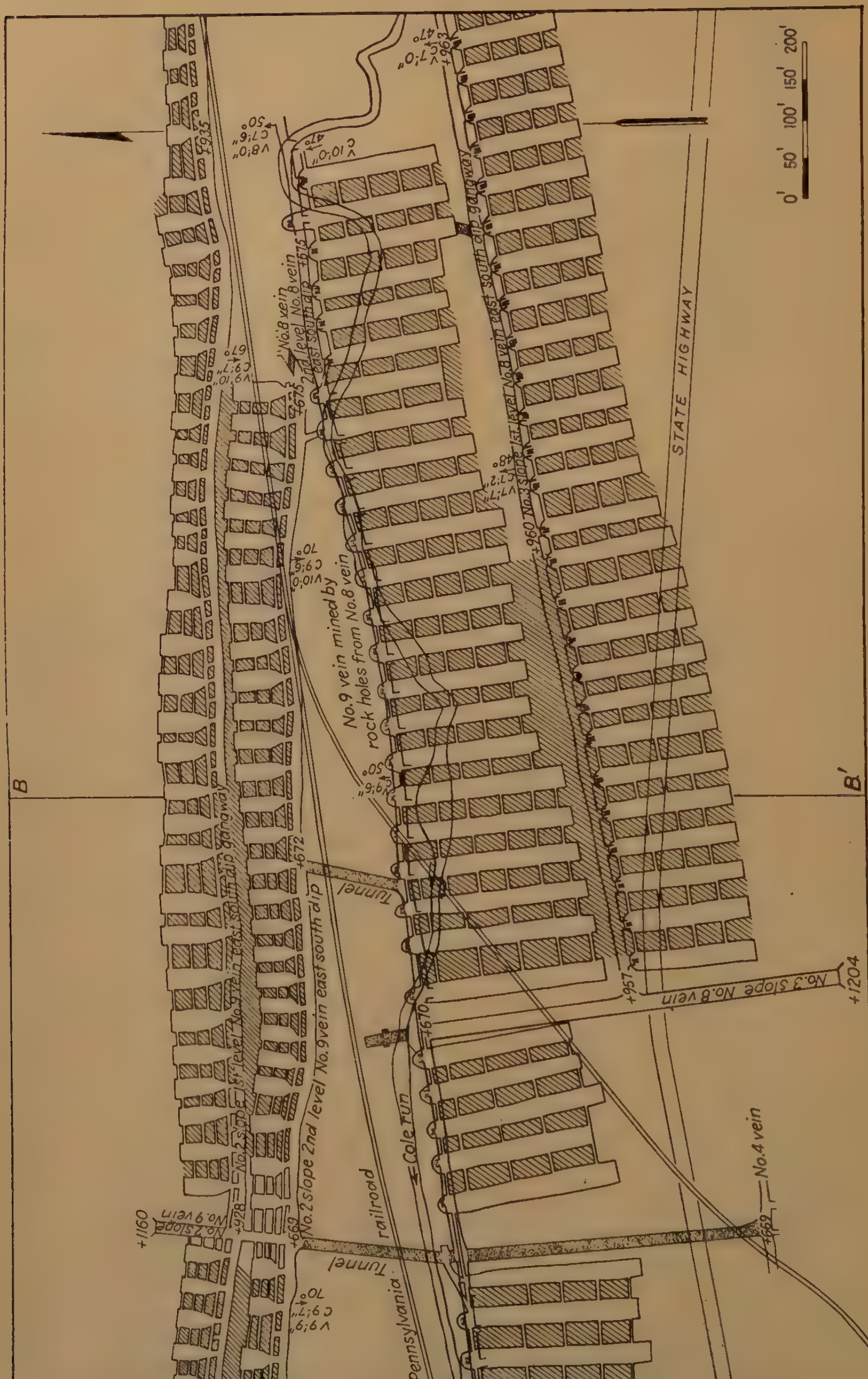

FIG. 5.—ROBBED AREA, NO. 9 SEAM, HICKORY RIDGE SECTION.

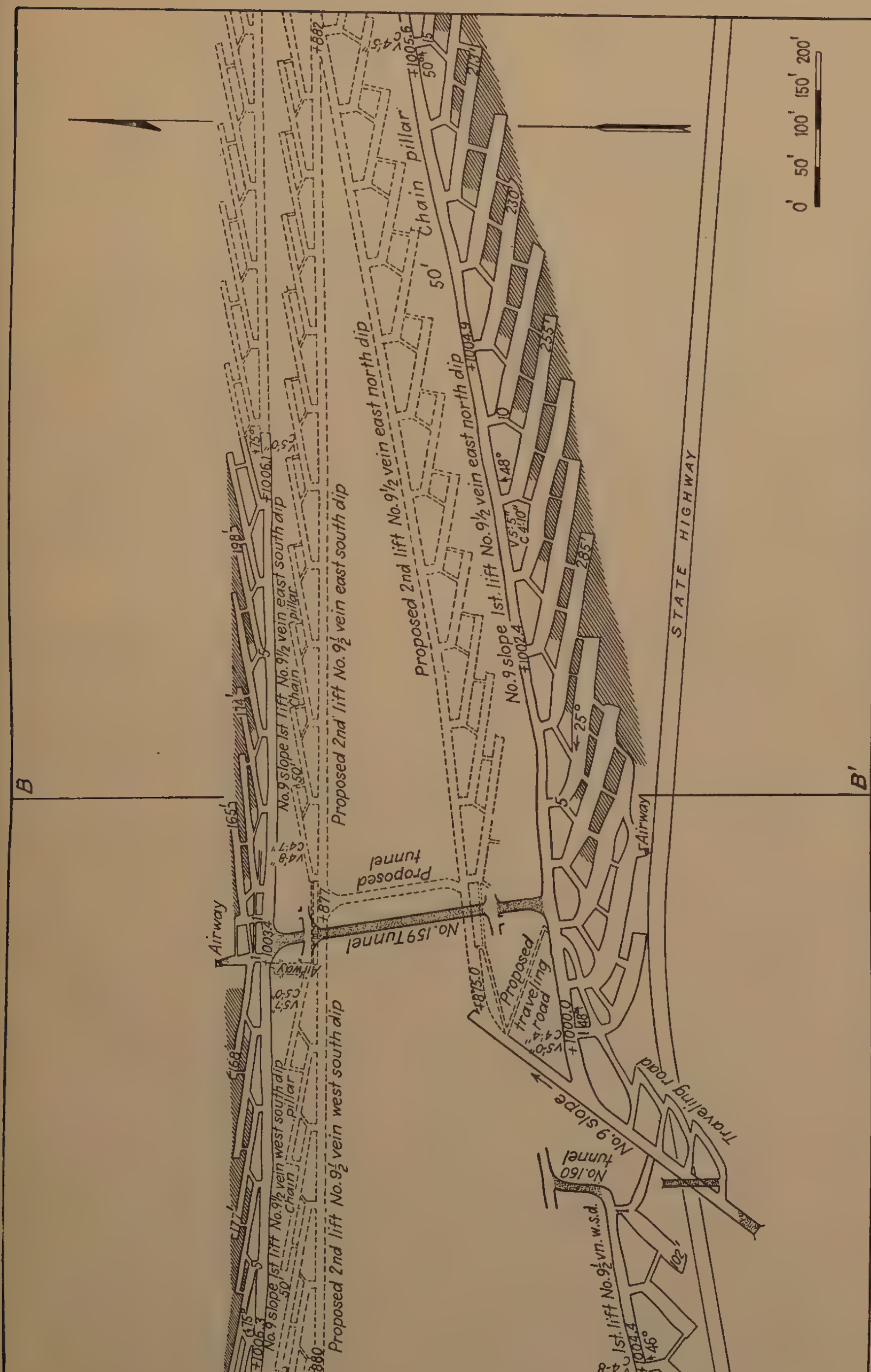

Fig. 6.—Mining method and proposed development No. 9½ seam, Hickory Ridge section.

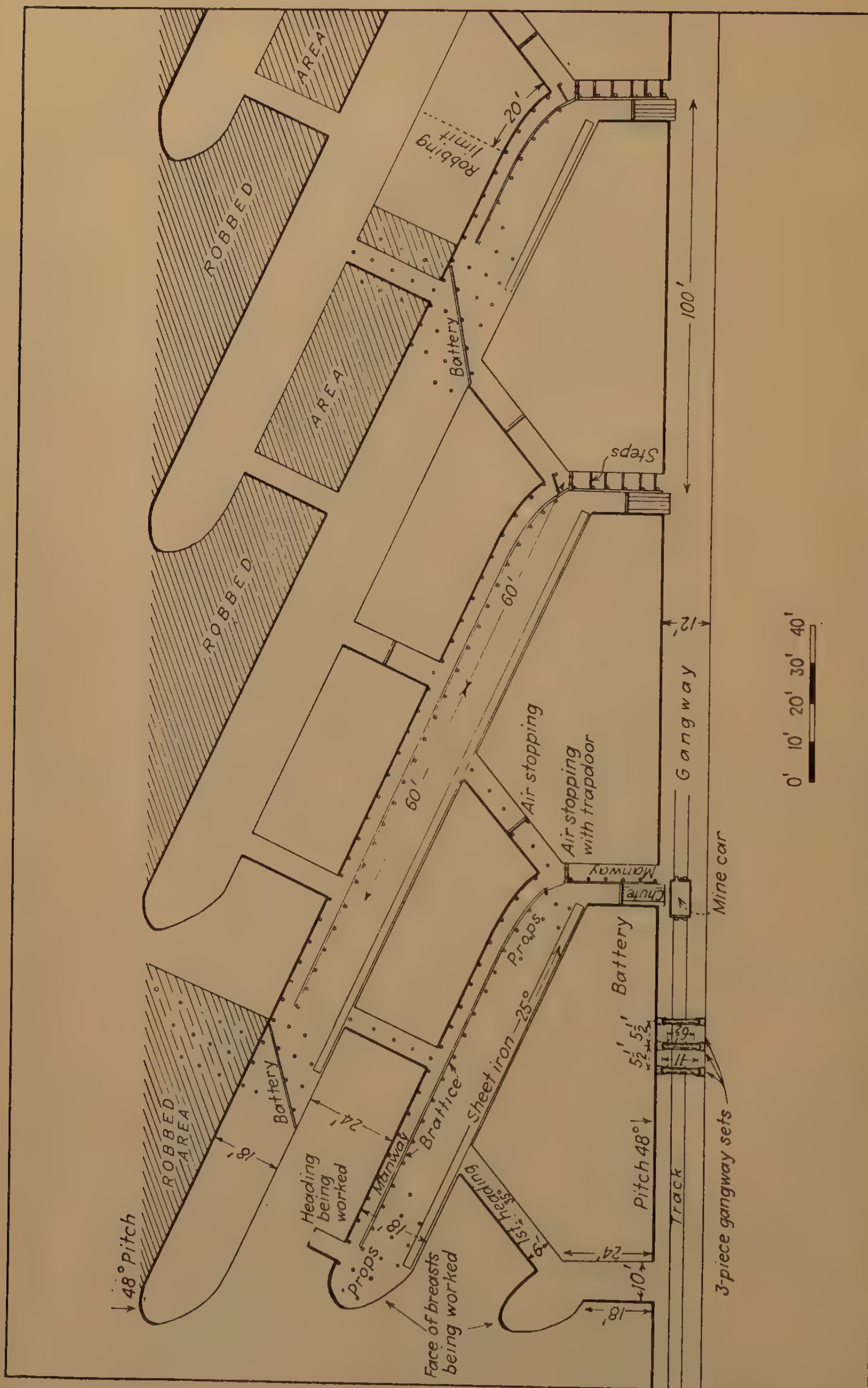

FIG. 7.—SLANT METHOD OF MINING.

across the pitch on 25°, in order better to control the clod and prevent it from falling and mixing with the coal, thereby increasing the yield of coal per car.

The "slant" method of driving the breasts across the pitch is shown in Fig. 7. Chutes were driven on 100-ft. centers off the gangway, 10 ft. wide, and were driven up 24 ft. with the pitch of the seam. From the face of each chute a heading 6 ft. wide was driven outby across the pitch on 35° to connect to the outside slant breast. The slant breast was then started inby at a point 18 ft. above the gangway on a pitch of 25° across the strike of the seam. Breasts were driven 24 ft. wide, except occasionally, when they were narrowed to 18 ft. because of the amount of top exposed and uncertain conditions of the top, caused by the draw of the mining and robbing of the lower seams.

As the breast advances, a single width of sheet iron 26 in. wide is laid along the low side of the slant breast, on which the coal will run to the chute. Four rows of props 8 in. in diameter are set in this slant. These props are securely hitched into the bottom rock and tightened against the top with large cap pieces and lashed together with liners. Each prop is lashed to the prop behind and also to the props on the left and right. If the roof is exceptionally bad and broken, laggings are placed above the lashers as a safety measure. Brattice boards are also laid across the foot of these props to provide footing for the miners while working at the face. A brattice to conduct the return air from the face to the nearest heading is carried on the third row of props. The miners going to and from the face of the breast travel between the brattice and high side rib. Headings are driven every 60 ft. at right angles through the pillar connecting to the outside breast for ventilation. The face of the slant breast also is driven on an angle to the sheet iron, so that the coal will have a minimum fall as it is blasted out. At times the coal has been found to have been drawn away from the top clod, or the top clod, which is 1 ft. 10 in. thick, to have been pulled away from the main top 2 to 8 inches.

Upon completion of the slant breasts, which extend from 150 to 280 ft., the pillar on the high side of the slant breast is removed. In bringing the pillar back, the retreating face is kept at the same angle as the heading, or at right angles to the breast, which is approximately 45° opposite to the line on which the slant is worked. As the pillar is removed the same method of timbering is carried on, including the lashing of props and the placing of brattice boards to afford footing for the miners. As the pillar is removed the top clod and roof break. This is kept from sliding down the pitch onto the miners or into the mined coal by liners placed across the props in the form of a battery. The pillars are removed to a point 20 ft. above the first heading, in order to gain the greatest possible production from the area being mined. Gangway stumps and remaining pillars are removed in the final robbing when the gangway is abandoned. All work is on double shift.

Loss in Mining by Breast-and-pillar Method with the Pitch

If the conventional breast-and-pillar method were used in the mining of these seams it would be virtually impossible to obtain a reasonable recovery of coal because of the top conditions. In this method, the breast would be driven up the pitch or at right angles to the strike of the seam. The breast would be driven 24 ft. wide and as the pitch is over 40° it would be necessary to carry a 3-ft. manway on each side of the breast. To form these manways, props would be placed every 6 ft., on which brattice boards would be nailed, forming a box into which the coal would drop as it was mined from the face. The additional coal mined would be run down the manway and loaded from the chute at the gangway into the mine cars.

As this drop would cause the coal to break into smaller lumps, the degradation would be very great; while in the slant method this amounts to almost nothing.

When the breast has been driven its full length, and before any attempt has been made to remove the pillars, the coal in the gob box is loaded into mine cars on the gangway. When a start is made to draw the coal in the breast, the top slate over the gob box, which has been held in place by the support of the gob or loose material, breaks and mixes with the coal. Often these pieces are large and block the gob box, thereby preventing the coal from dropping to the point where it is to be loaded into the mine cars. In many cases this coal cannot be recovered, owing to the cost and danger of additional top breaking while an attempt is being made to reclaim it.

In the standard method of removing pillars, a pillar hole or chute 8 ft. wide is driven off the gangway in the center of the pillar. This is driven in the same manner as the breast, manways on both sides of the gob box being carried in the pillar hole. When the pillar hole has been advanced to the same height as the face of the outside breast, the remaining pillars on both sides are shot off and loaded out of the pillar hole into cars on the gangway. By this method a large amount of slate is loaded with the coal, as the top slate breaks as the pillar is removed.

In another section of this field Nos. 8 and 9 seams were mined and robbed, the pitch being approximately 80°. The coal was mined in two lifts of 300 ft. each, and in consequence the surface caved. Later an attempt was made to mine the No. 9½ seam, which was separated by only 30 ft. of strata from the No. 9 seam. It was found on the first lift, or lift closer to the surface, that the entire seam had fallen into the void left when Nos. 8 and 9 seams were mined. In attempting to mine this No. 9½ seam from the lower lift, it was found that breasts could be worked for 250 ft. but beyond that point the strata were broken and too dangerous to mine. The broken strata from above had fallen into the void left by removing Nos. 8 and 9 seams above the second lift and held the strata under the No. 9½ seam in place to the point at which the breasts were stopped.

Experience has proved that thin seams can be and are being mined in the western field over large areas that have been mined and robbed, although the mining would be less expensive if the lower seams had not been mined, for this has displaced the strata and in some places crushed the coal.

In observing the effect of mining flat seams over robbed areas in the northern Anthracite field, we have tried to check on certain conditions to obtain some definite information to guide us in forecasting our development program. These conditions are:

1. Whether the overlying seam to be mined is solid or pillar coal.

2. Respective thickness of the overlying and underlying seams.

3. The interval of rock between the seams; also the nature of rock.

4. Depth of the seam below the surface.

5. Acreage to be mined.

6. Time interval between the robbing of the underlying seam and mining of the overlying seam.

7. Dip of the seams.

8. Recovery per foot-acre compared with mining over solid or pillar areas.

A typical example of what has been encountered at one of the northern collieries of the Susquehanna Collieries Co. was the first and second mining of an area of 20 acres of the 8-ft. thick Twin seam overlying a completely mined area of the 5-ft. thick Top Ross seam, with a rock interval of 70 ft. consisting of medium hard sandstone and sandslate. Fifty feet under the Top Ross, the 7-ft. Bottom Ross seam had also been completely mined. The dip of these seams was 10°.

The time interval between the completion of mining in the underlying seam and the beginning of mining in the upper seam was 6 years. No difficulties in mining were experienced, and maintenance cost and recovery were the same as in sections where the coal below had not been disturbed.

At the present time the Susquehanna Collieries Co. is developing and mining a 100-acre section of the Forge seam. About half of this area overlies the 8-ft. thick Twin seam robbings. The mining has extended over 8 acres of Twin robbing and no adverse effects have been noticed. The rock interval between the Forge and Twin seams ranges from 140 to 180 ft., with a dip of 15°.

In a near-by colliery a 30-in. seam of solid coal was extensively mined by scraper loaders; most of the mining being done over a robbed area of an 8-ft. seam lying 70 ft. below it. In this case it was noticed that when the mining advanced over the robbed area the coal was more easily shot and more lump coal was obtained; also, occasional cracks were found in the bottom rock. The total recovery of this 30-in. seam was about 85 per cent.

In connection with this case it is worth while to mention the effect of the mining on the surface overlying the area. Levels have been run over the surface at four different periods—in 1894, in 1917, in 1920 and in 1929. The bench mark used was outside the coal measures.

The center of maximum subsidence was at a street intersection and the levels showed that for the period 1894 to 1917 the subsidence was 6 ft.; from 1917 to 1920 an additional 3 ft.; from 1920 to 1929 a subsidence of 5 ft. more, or a total of 14 ft Underlying this point 10 seams with original thickness of 55 ft. have been mined and most of them robbed. The 30-in. seam cited was the third seam from the surface at a depth of 350 ft. This seam was mined in the period between the 1920 and 1929 surveys, so that a 9-ft. surface subsidence had already occurred before mining started. The surface seam lies 250 ft. below the surface and the lowest seam is 1140 ft. deep.

The thicker seams had been partly mined over and some completely robbed during the 75-year period preceding the development of most of the thin seams. It is believed that the subsidence taking place in the seam in question was so gradual and spread over such a large area that it permitted the successful mining of the seam.

An attempt is being made to mine a 3½-ft. thick solid Rider seam over a first-mined 8-ft. thick Top Ross seam. The rock interval of sandslate between these seams varies from 4 to 6 ft. in thickness.

The Top Ross seam was mined about 40 years ago with road breasts driven 24 to 30 ft. wide. These road breasts were not driven to line but on grade that would suit mule haulage, and since the dip of seam is irregular (about 5°) the road breasts are not parallel, and no attempt has been made to columnize the mining. To date the breasts have been advanced without difficulty.

At another Susquehanna mine a 4-ft. seam had been completely mined underneath a Bottom Ross pillar area. The rock strata between these seams consisted of hard sandstone 90 ft. thick. From previous experience no trouble is anticipated in robbing the Bottom Ross pillars, but these pillars proved to have been subjected to a heavy squeeze and some small caves were encountered. Reopening and maintenance costs in this area were slightly more than average, but the average recovery for this seam was obtained.

At this same mine, 30 acres of 5-ft. thick Forge seam were mined over the completely mined Twin seam, which was 8 ft. thick. The rock strata between seams was 250 ft. and the pitch 10°. This Forge area contained both solid and pillar coal. In first and second mining of the solid coal no difficulties were encountered; but when robbing the pillar coal it was found that in nearly every breast the top slate, about

3 in. thick, had caved. These pillars were recovered by splitting the pillar, using shaking chutes to handle the coal. In other sections where the underlying Twin seam had not been robbed the Forge breasts remained in fairly good condition.

It should be noted that in no instance have overlying seams had to be extracted where underlying seams 15 ft. thick or over have been completely mined.

From experience in the western end of the northern Anthracite field, the following conclusions have been drawn:

1. Better results will be obtained when the seam to be mined, overlying a completely mined area, is solid coal rather than pillars; since any subsidence taking place does not affect the immediate roof of the solid coal as much as it does a pillar area.

2. If the rock strata between seams is 100 ft. or more and the underlying seam does not exceed 10 ft. in thickness, successful mining can be accomplished in the overlying seam in most cases.

We have not been able to arrive at any conclusions as to the effect of the depth of mining below the surface, the time interval between the mining of overlying and underlying seams, or the acreage to be mined.

Mining-machine Bits—Experience and Practice

By A. Lee Barrett,* Member A.I.M.E.

(New York Meeting, February 1940)

So commonplace that they are seldom noticed, mining-machine bits have a definite and important bearing on the cost of coal production. At the average mine many thousands of bits are used during the year. The actual cost represented by the purchase of the bits used, as well as that of sharpening and handling them, represents a considerable sum of money. From the point of view of indirect costs they are important, as the lack of good bits at a mining machine can seriously impair production. Good bits tend to ensure an adequate supply of coal.

Classes of Bits

Mining-machine bits may be divided into three general classes, based on shape and method of use: (1) chisel, (2) pick-point and (3) one-use. The chisel and pick-point bits are shown in Fig. 1. The so-called "chisel" bits were made of steel of various sizes and were sharpened to a wide chisel cutting edge. Few of this type of bit are used, except in places where coring is very bad. Sometimes chisel bits are used with standard bits to help cut out the core.

The "pick-point" bit is widely used today. Normally it is made from ½ by 1-in. steel, is about 4½ in. long when new, has a rake angle of about 25°, a front clearance of about 23°, a side clearance of approximately 8°. There are many variations in the shape of this bit, depending on the ideas of the individual that may be dressing the dies of the bit machine. This bit has been generally used for many years and at present, in one form or another, is by far the most popular bit used in coal mines. This bit

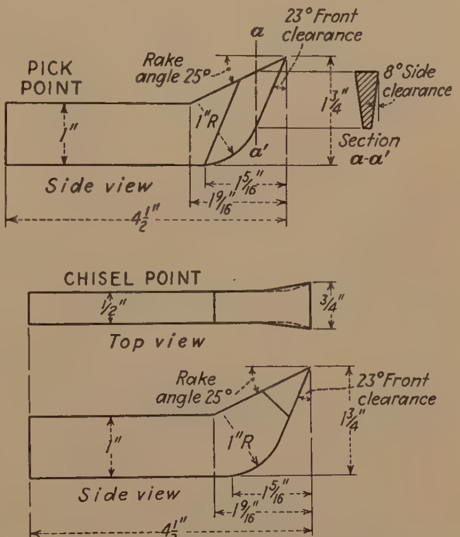

Fig. 1.—Chisel and pick-point bits.

usually is made from 0.85 carbon steel, although in recent years some alloy-steel bits have made their appearance. At present, carbon-steel bits are frequently tipped with hard facing materials.

A third class of bits might be called the "one-use" bit; that is, a small bit that is used once and then thrown away. These bits have two or three points, are usually made from 0.85 carbon steel and sometimes from higher alloys. Recently some of them have been tipped with hard facing material. Most of them were used at first with special

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* Pittsburgh Coal Co., Library, Pa.

chains but in recent years several bits of this class have been made for use in standard cutting-machine chains with adapters.

Characteristics for Best Performance

Regardless of the class to which the bit belongs, there are definite characteristics required of all bits for best performance. Perhaps the most important of these is low cost, which should take into account the purchase price of the new bit and the cost of redressing, together with the handling of the bit in and out of the mine. Compromises necessarily must be made if it is found that the increased first cost of the bit, its redressing and handling exceed the saving to be derived from the greater length of service of the better bit.

A good mining-machine bit should have high performance with respect to the number of square feet or tons cut per bit. A high-grade bit that cuts many square feet per sharpening gives higher tonnage per cutting-machine shift and as a consequence permits lower cutting cost. This is partly the result of time saved by reduced bit changes. If the cutting machine is working in a loading-machine territory where the tonnage is not set by the cutting machine but by the loading machine, high-performance bits make the operator and his helpers available for other work. The higher the performance of the bit, of course, the fewer the bits that must be sent in and out of the mine. This reduces handling cost, makes possible the installation of a more efficient bit-handling system and tends to reduce the loss of bits in handling, both at the bit-sharpening plant and at the cutting machine itself. High-performance bits usually provide better cutting-machine performance; that is, it is possible for the machine to cut material that cannot be cut with lower grade bits. For instance, rolls, clay veins or spars may be cut with a really good bit whereas an ordinary bit might not cut them or might do so at the sacrifice of maintenance cost on the mining machine. The ability to undercut rolls, spars, and clay veins reduces the cost of removing these materials, of course. A high-performance bit that can really take abuse reduces machine maintenance and power in unusually difficult cutting. If the cutting is so difficult that an ordinary bit would fail and the machine operator continues to cut the material, adding extra load to the power system, a severe strain is imposed on the machine, which often causes mechanical failures. This does not occur when a bit is available that can readily cut the material. This statement, however, should not be construed to mean that a high-performance bit decreases either maintenance or power cost in normal cutting practice, as it definitely does not do this. A common opinion is that an improved bit material or better tipping will reduce maintenance cost and power consumption. Extensive tests made with various types of bits at various mines do not bear out this contention in any way. If a bit has the proper shape to do a good job of cutting the power consumed when that bit is new and when it is worn to the point of changing is about the same, regardless of the material from which the bit is made. It follows, then, that increased power consumption and maintenance cost on cutting machines is a result of the dullness of the bit, if bits are changed when they reach a certain state of dullness, power consumption and maintenance cost should be the same without regard to the type of bit material used. It would be true that if a superior bit material could be discovered, permitting the manufacture of an improved shape of bit, the maintenance cost would be reduced. Were an alloy available for bit manufacture with sufficient durability and hardness, it would be practical to change bits when they were in a relatively sharp condition, thus gaining a reduction in power and maintenance costs. At the present time, unfortunately, no materials with such characteristics are available.

In studying the performance of bits, the variation in the characteristics of the strata being cut is significant. Where cutting is being done in the coal, a wide variation in characteristics exists. Abrasion, brittleness, toughness, etc., all are important factors influencing the cutting action of the bits. Coals containing streaks of "bone" and particles of pyrite finely disseminated throughout the seam often impose a severe grinding action on bits, cutting chains, and other mechanical parts with which they may come in contact. Maintenance and bit life are influenced by this wear and tear caused by encountering such foreign bodies as sulphur balls, clay veins, spars, etc. All of these factors must be taken into account when comparing various bits. In the selection of a cutting-machine chain and its bit, all of these influences should be carefully studied and the choice of both bit and the accompanying chain should be based on the type of cutting action that will yield the highest economical results under the existing conditions. In good cutting practice, each bit should break the coal between its point and the channel cut by the preceding bit, ensuring a minimum of actual cutting. The minimum number of bit positions compatible with the characteristics of the coal being cut should be used. This depends usually on coring; only a sufficient number of positions should be used to cut a clean kerf. The number of positions ranges between five and ten with nine and ten-position chains, in prevalent use at the present time. In coal of certain characteristics the number of positions in use might be reduced if enough attention were given to the problem.

Because of the variation in coal characteristics, the chain lacing yielding the most satisfactory performance in a certain coal field or mine may not necessarily apply with equal efficiency in other sections or in adjacent mines. For coal of uniform characteristics, which does not have a tendency to "core," the "flying wedge" (Fig. 2) is probably the best type of lacing. In hard-structure coal, good results may be obtained from the flying wedge with extra top and bottom bits. Where there is a variation in hardness from the top to the bottom of

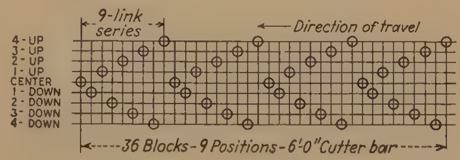

FIG. 2.—FLYING-WEDGE ARRANGEMENT OF CUTTING CHAIN.

the kerf, satisfactory operation may be obtained through the use of some modification of the wave type of lacing. Frequently the coring problem can be overcome by a special type of lacing.

Considerable experimentation has been done with respect to the proper bit angle. Casually it would appear that a bit would yield most satisfactory service in cutting coal if it approached the coal face in a perpendicular position. This is not true, as a balance between side and vertical pressures must be maintained to obtain good cutting action. When outside bits make an angle of approximately 45° to the vertical the tendency to squeeze the bits together is to be overcome. A number of chains arranged with a reduced bit angle were installed at one of the properties in 1937, but a study of the performance revealed that this arrangement definitely reduced the cutting ability of the machine and increased the power consumption at least 30 per cent.

Hand Sharpening Versus Machine Sharpening

When the pick-point bit was first developed it was hand-forged, usually from a 0.85 per cent carbon steel. In many places, this is still the practice, and where a good blacksmith is available it is doubtful whether the use of the average bit-sharpening machine can show any saving in cost over the hand-forged bit. A good blacksmith will do a better job of heat-treating than can be done with a bit-sharpening

machine, as the bits are usually quenched from the forging heat. Further, the hammering of the bit during hand-forging tends to compress the grain, yielding a better structure of the tip. A comparison of the cost per 1000 sq. ft., as shown in Table 1 for hand-sharpened bits and Table 2 for machine-sharpened bits, will indicate that there is no appreciable cost advantage for machine-sharpened bits. At most large mines approximately five years will be required for a bit-sharpening machine to return the investment when based solely on the cost of bits. During that time there is certainly some extra labor cost because of the extra bits that must be transported and the extra number of bits that must be changed in the cutting-machine chain. Alloy-steel bits are usually not forged by hand because of their toughness.

Mechanical sharpeners usually fall into one of three classes, the roll type sharpener, a sharpener in which a bit is forged by squeezing it between two dies and a sharpener in which the bit is actually hammered. Each type has its own particular advantages and disadvantages. Although the roll-type sharpener has a very satisfactory maintenance cost and a relatively high speed of operation, it leaves a fin along the whole cutting edge of the bit, particularly if it is not in perfect mechanical condition, and as this cutting edge is the point where it should be sharpest, the bit must be ground if best performance is to be attained. The squeezing type of forging machine is also satisfactory from both output and maintenance points of view, but this type of machine does not readily straighten a bit that gets out of shape, particularly on the cutting face. Neither of the other machine types mentioned yield satisfactory results when sharpening bits of tough alloy steel.

The recently developed hammer type of bit sharpener employs a hammering action that reduces the size of the grain in the steel, yielding a better grain structure. It is highly satisfactory throughout, and by hammer forging bits may be sharpened at lower temperatures than by other methods. Tough alloy materials may be handled with ease on a hammer-type machine and when the size of the machine and the power requirements are taken into consideration, this type of machine yields the best over-all results.

The heat-treatment of machine-sharpened bits as practiced at Pittsburgh Coal Co. mines consists of running the bit down a conveyor, which allows the forging heat to run back out into the tip of the bit, after which the bit is dropped into a container having water mixed with some tempering material, which delays the quench. It is recognized that this is a very poor form of heat-treatment and that the grain structure of the bit is poor. However, extensive experiment carried on over several months at two mines indicate that the cost of cooling, quenching and drawing, does not justify itself in increased bit performance.

In a test run at one of the mines (mine L), covering a period of one month, the whole mine was supplied with re-heat-treated bits; 36,682 bits were used in cutting 58,267 tons, an improvement of 25 per cent on the number of tons cut per bit was effected, but unfortunately there was an increase of some 40 per cent in the cost per bit, so that the cost per ton was higher. A similar test was run for several months at mine K, with virtually the same results. These bits were heated, forged, cooled in air, reheated, quenched and drawn. The bits used at mine K were oil-quenched while bits in operation at mine L were water-quenched. The explanation of these phenomena can be seen in Fig. 3, which shows the limit of hardness to be reached by cold water quenching and by annealing carbon steel at various carbon contents. Since machine-sharpened bits cannot be quenched in cold water, the maximum hardness reached by 0.85 carbon steel, when properly heat-treated, is approximately 500 Brinell.

Hardness usually reached by machine-sharpened bits, which are quenched on the forging heat, is 400 to 450 Brinell. Both of these hardnesses are limited by breakage.

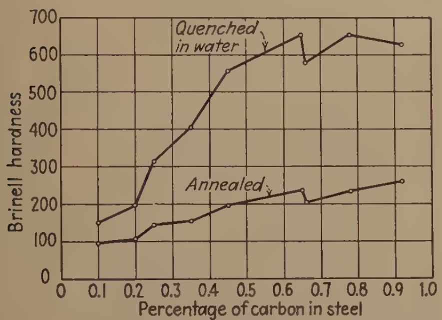

FIG. 3.—LIMITS OF HARDNESS FOR CARBON STEEL.

No matter how poorly an 0.85 carbon bit is tempered, it is a rather hard bit capable of considerable cutting. In connection with machine-sharpened bits, it is important that the forging temperature should not be too high. Forging bits at moderate temperatures produces good grain structure and does not burn the steel.

TESTING PERFORMANCE OF BITS

In testing the performance of bits, considerable care must be exercised if significant figures are to be obtained. All performance characteristics must be based on the square feet of undercut and the best way of comparing power performance is on the basis of watt-hours per square foot. The variation in coal height makes comparative performance based on tons misleading. In order to determine the power consumption accurately an integrating watt-hour meter must be used, instead of a graphic watt-meter, or voltmeter and ammeter. Preferably an arrangement consisting of an integrating watt-meter and a graphic watt-meter should be used with records of both meters on the same chart.

Fig. 4 is a photograph of this test equipment as used by the Pittsburgh Coal Co., together with the small truck provided for convenient transportation. A voltmeter should be carried to check the voltage. Fig. 5 shows a graphic chart with integrated record on the edge.

FIG. 4.—TEST EQUIPMENT WITH SMALL TRUCK USED FOR TRANSPORTATION TO WORKING PLACES.

Care must be used in interpreting test data after they have been obtained. The operator can introduce considerable variation in the performance of a cutting machine by the method of handling the machine, and if there is variation in the cutting characteristics in the portion of the coal being cut, the position in which the machine is operated will greatly affect the power consumption. Table 1 is a good illustration of this variation. This table covers the performance of tipped bits in 13 sections of mine L. The bits were all made of the same steel and were all tipped with the same material. The tests cover a period of 18 working shifts. A large number of tests should be averaged before any decision is reached as to cutting capacity of a given bit or power consumption of the machine.

PERFORMANCE DATA

The following performance data are based on experiments that have been run from time to time at the various mines of the Pittsburgh Coal Co., and actual daily performance records for the year 1939 (to

Dec. 1). In comparing the various types of bits involved, mines A, B, F, G, H, I and J have similar cutting characteristics. Mines

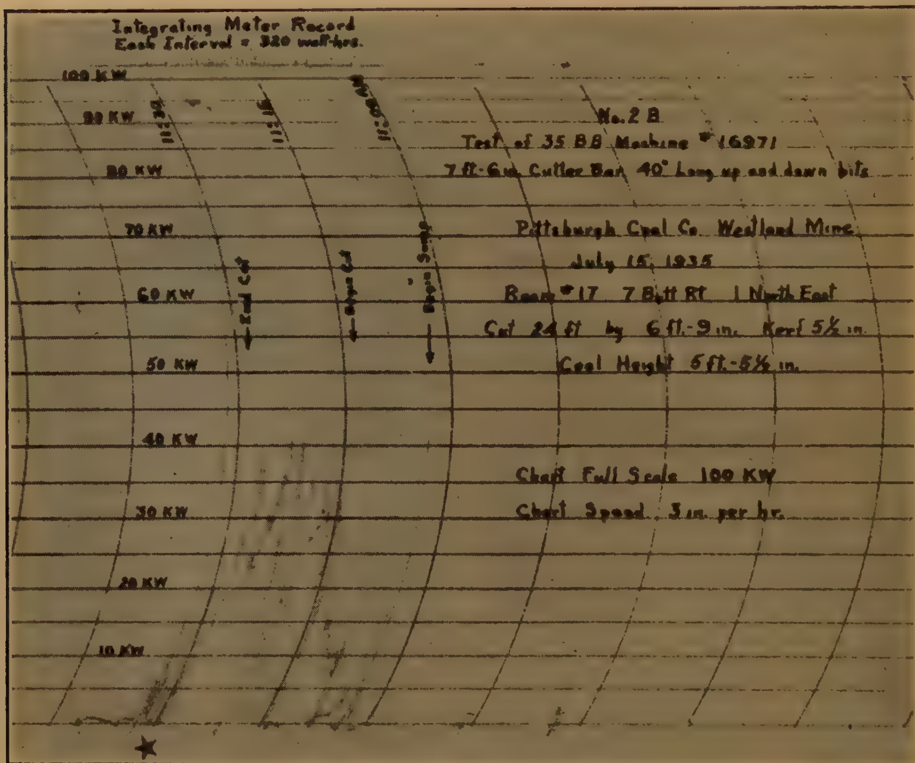

FIG. 5.—GRAPHIC CHART WITH INTEGRATED RECORD ON EDGE.

D and E are similar, and mines K, L, M, N and O are similar.

Hand-sharpened carbon-steel bits of a standard pick-point shape made of 0.85 carbon steel are in use at five Pittsburgh Coal Co. mines. Performance of these bits is shown on Table 2*A*. The life of a bit in terms of the number of times each bit may be sharpened is relatively high at mines where the bits are hand-forged because of the superior heat-treatment given to hand-sharpened bits. The cost of hand-sharpened bits at mine A is high because few bits are used at this mine, which is producing coal chiefly from retreat workings. Also, apparently, more care is taken with the treatment, and this, although it improves the performance of the bit, does not reduce the cost per ton. Mine B, which is very near mine A, has better performance in this respect. No attempt is made in any of the tables in this paper to show average results except in regard to sharpenings per new bit. There is such a wide variation in cutting characteristics in the various mines involved that an average would be meaningless.

Table 2*B* indicates the performance characteristics of machine-sharpened 0.85 carbon standard pick-point bits at eight mines. The cost varies with the practice of sharpening at the individual mines and with the number of bits sharpened at the mine. The cost per square foot cut varies inversely with difficulty in cutting as measured by the square feet cut per bit.

TABLE 1.—*Stellite Bit Test, Mine L*

Location of Cutting Machine	Tons per Bit	Location of Cutting Machine	Tons per Bit
2 face north 15 butt left	3.18	13 face south 7 butt	1.96
2 face north 17 butt left	5.07	13 face south 11 butt	1.23
5 face north 7 butt	5.20	13 face south 21 butt	2.71
5 face north 19 butt	5.50	13 face south face entries	1.32
Main east 1 butt N	2.35	3 face south 25 butt	2.12
Main east 1 butt S	2.51	3 face south 29 butt	1.10
Main east 3 butt S	2.19		

TABLE 2.—*Performance of Bits*

Mine	Total Tons Cut	Tons Cut per Bit	Bits Used			Sharpening per New Bit	Cost			Watt-hours per Sq. Ft.	1000 Sq. Ft. Cut	Sq. Ft. per Bit
			New	Sharpened	Total		Per Bit	Per Ton	Per 1000 Sq. Ft.			
A. Hand-sharpened 0.85 Carbon Bits												
A	214,000	6.52	500	32,300	32,800	64	$0.0235	$0.0036	$0.93	30	830	25.0
B	368,000	3.55	2,450	101,400	103,850	41	0.0104	0.0029	0.71	40	1,520	15.0
C	206,000	1.82	1,540	111,800	113,340	72	0.0086	0.0047	0.84	40	1,160	10.0
D	107,600	1.80	1,000	58,730	59,730	59	0.0082	0.0046	0.98	40	500	8.4
E	566,900	1.70	2,725	330,800	333,525	120	0.0062	0.0037	0.69	45	3,000	9.0
			8,215	635,030	643,245	77						
B. Machine-sharpened 0.85 Carbon Bits												
F	736,300	4.18	4,000	172,100	176,100	43	0.0088	0.0021	0.43	30	3,630	20.6
G	367,500	4.01	3,000	88,600	91,600	29	0.0043	0.0011	0.27	30	1,440	15.7
H	589,500	3.90	3,000	148,000	151,000	49	0.0083	0.0021	0.53	30	2,360	15.6
I	467,600	3.80	2,100	120,900	123,000	57	0.0075	0.0020	0.46	30	2,000	16.2
J	658,200	3.07	8,000	214,100	222,100	27	0.0075	0.0025	0.59	40	2,800	12.6
K	922,700	1.27	18,500	709,900	728,400	38	0.0047	0.0037	0.77	55	4,450	6.1
L	1,025,700	1.15	27,500	862,400	889,900	31	0.0042	0.0036	0.77	55	4,840	5.4
M	614,400	1.01	18,900	587,000	605,900	31	0.0042	0.0041	0.83	55	3,080	5.1
			85,000	2,903,000	2,988,000	34						
C. Tipped Bits												
N[a]	924,000	3.12		268,000	268,000		0.0102	0.0032	0.72	55	4,210	14.2
O	166,000	2.91	1,000	55,900	56,900	56	0.0078	0.0027	0.54	55	835	14.7
			1,000	323,900	324,900	56						
D. Above Mines When on Standard 0.85 Carbon Machine-sharpened												
N	1,248,000	1.38	49,200	853,400	902,600	17	0.0057	0.0041	0.90	55	5,700	6.3
O	377,600	1.04	2,500	363,000	365,500	15	0.0059	0.0057	1.14	55	1,890	5.2
			51,700	1,216,400	1,268,100	16						
E. One-use Bits (Tipped)												
	120,300	13.31	9,038		9,038		0.0475	0.0036	0.71	55	604	67.0
F. Alloy-steel Bits Machine-sharpened												
H	1,427	11.10	128		128	50	0.007	0.0006	0.16	30	5.22	45.0
O	732	4.44	167		167	50	0.007	0.0016	0.32	55	3.73	22.0

[a] Information on new bits not available at this mine. Costs calculated on basis of sharpening cost only.

As mentioned before, these bits are all quenched from the forging heat.

Apparently one of the weaknesses of the 0.85 carbon steel bit is that when it heats in

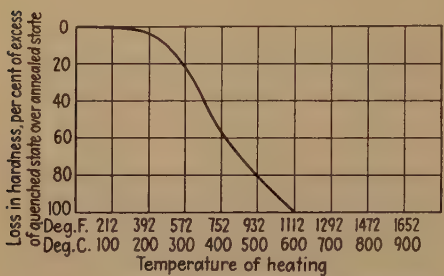

FIG. 6.—LOSS OF HARDNESS IN TEMPERING STEEL.

the coal-cutting operation it is drawn back from its original hardness. Fig. 6 shows the drawing effect on 0.85 carbon bit steel. When a temperature of over 400° is reached its hardness begins to fall off rapidly (Fig. 7) and the ability of the bit to continue to perform satisfactorily is considerably hampered. Particularly in very hard cutting does this characteristic show. Many bits are found that apparently have reached a very high temperature during the cutting operation and are forged to all sorts of shapes by the coal. This is one of the weaknesses that the tipped bit and the alloy-steel bit tend to overcome.

Table 2*C* shows the performance of tipped bits at two Pittsburgh Coal Co. mines. For the sake of comparison there is included in this table the performance of machine-sharpened bits at the same mines during an earlier period. The table gives a good comparison of cutting performance as well as cost. (Unfortunately, the figures on new bits used in mine N are not available and as a consequence the sharpenings per bit at this mine cannot be shown. Also, the cost is figured on resharpenings only and does not include new bit steel. However, this should amount to only about one mil, so that the cost is not far out of line.) Tipped bits, in general, cut from two to three times as many square feet per bit as do the machine-sharpened 0.85 carbon bits. This average performance appears to be well borne out by a large number of tests run at various Pittsburgh Coal Co. mines.

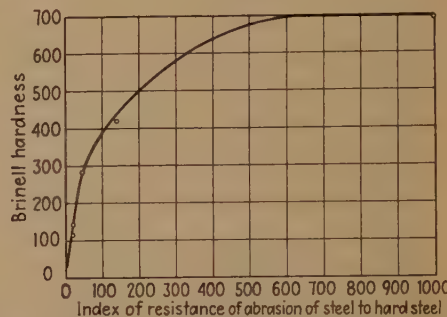

FIG. 7.—WEARING QUALITIES OF HARD STEEL.

In certain sections very much better performance can be achieved. It is probably true that in certain parts of the country where cutting conditions are easier than in the Pittsburgh district very much better performance can be obtained. It is questionable whether the bit life shown at the mine O can be considered to be conclusive, because of the relatively small quantity of bits involved and also because just prior to 1939 and during 1939 several of the machines in the mine were being changed to use a one-use type of bit. This made available a large number of bits that had been used on these cutting machines. In general, it is indicated that when bits are tipped with hard facing material, bit life is likely to be shorter than that of ordinary machine-sharpened bits. These bits are machine-sharpened and before tipping are ground so that tipping material will flow into place evenly. Standard 0.85 carbon steel in the standard pick-point shape of bit is used. No bits are reground, they are either redressed or sent back into the mine to be used again.

Table 2*E* shows the performance of one type of one-use bit at mine O. The table is based upon extensive tests on several machines for the year 1939 (to Dec. 1). It will also be observed that the performance of

this bit has been particularly good. The bit in use is a two-pointed bit tipped with a hard facing material of the carbide type and is used in a standard chain with adapter. Bits that come out of the mine with life left in them are ground and put back in service. In spite of the excellent performance of this bit, there is nothing unusual about the cost per square foot of undercut, but there are many economies resulting from fewer bits to transport and from fewer bits to change in the cutting-machine chain. For the past several years many tests have been run on various types of one-use bits. Most of them have been made of heat-treated carbon or alloy steel. The performance of most of these bits, in square feet cut per point, has been about the same as that of the tipped bit made from 0.85 carbon steel and of the standard pick-point shape. Because of the high original cost of these bits, a good cost per square foot of cut was never achieved with any of them. The one-use bit made of alloy steel properly heat-treated and tipped seems to be well worth watching, however.

Extensive tests have been run in the past few months and are now being run on alloy-steel bits sharpened on the hammer-type machine. The particular alloy steel being used in these bits has a tensile strength of over 300,000 lb. per sq. in. This steel air-hardens to 62 Rockwell, which corresponds to about 625 Brinell, has an elongation of approximately 20 per cent and appears to be very tough. The cost of this material is 15¢ per bit. In Table 2*F*, showing performance of this bit to date, sharpenings per bit has been estimated at 50, but performance indicates that this may well run over 100. These bits may be sharpened at the rate of at least six per minute, either when being resharpened or when being forged from original bar stock. Performance has been slightly less than two times that of tipped bits in tests run so far. On the basis of 50 sharpenings per bit, the cost is encouraging. If the sharpenings per bit prove to be as high as the present tests indicate, an even better cost per square foot will be obtained. The weakness of this kind of bit is that it must be sharpened in a special hammer-type machine. It is entirely too tough to forge by hand or in any machine on the market other than the hammer type. In view of the fact that the price of the furnace and machine is something over $2000, it is obvious that a large number of bits must be handled by the machine in order to justify its cost. It is probable that where a relatively large number of mines are grouped in an area, such as are those of the Pittsburgh Coal Co., one or two central bit-sharpening plants would provide a cost for alloy-steel bits that would result in a very satisfactory cost per square foot of cut. Where this is not possible, it is probable that either the ordinary machine-sharpened 0.85 carbon bit or the tipped bit will do a better job, when all of the costs are considered.

Conclusions

Conclusions that might be drawn from experience to date would run somewhat as follows: The best bit is one that breaks the coal and does not cut it. If bits are changed when they begin to get dull, maintenance cost and power cost should be the same for any good bit. These factors are not noticeably affected by the number of square feet cut by the bit. The only satisfactory basis for comparing bits is by kilowatt-hours per square foot of undercut. Owing to the wide variation in cutting characteristics and in the handling of cutting machines, many tests must be run before an average can be reached that will be representative. The comparative performance of hand-sharpened and machine-sharpened bits should be investigated before a bit machine is installed. Before improved heat-treatment is given to bits, relative costs should be ascertained. Frequently the improved performance of the bit does not justify the cost of making the improvement. Experience to date indicates that even the best one-use

bit cannot be justified except on the basis of saving in bit-changing time and saving due to reduced transportation difficulties. The introduction of an improved hammer-type bit-sharpening machine and a high-grade alloy steel, which is air-hardening, indicates that a bit of this type may well lead the field wherever a large number of bits are involved. Bit experience at one mine does not necessarily foretell experience at any other mine. General conclusions should not be reached from limited local tests. Only a careful long-time test will justify definite conclusions.

Methods of Borehole Lining

By John S. Johnson,* Member A.I.M.E.

(New York Meeting, February 1941)

The purpose of this article is to describe several types of borehole lining in common use, and especially to offer a relatively new means of reducing the expense of maintaining boreholes where they are subjected to corrosive action of water flowing either inside or outside of their casing.

In the Anthracite region of Pennsylvania, boreholes are used for many purposes, the principal ones being: openings between the surface and underground workings through which are passed steam, water and compressed-air lines, ropeways from surface hoists to underground rope haulages, conduits for electric power and signal lines, water-discharge lines from underground pumping stations, and inlets to mine workings for flushing material.

Linings for Boreholes

The inadequacy of steel pipe for use in boreholes has long been recognized. Various substitute materials have been tried, including screw-joint cast-iron pipe (Fig. 1).

Cast-iron pipe, while very effective as a discharge line for acid water (mine water) being pumped to the surface, is not so effective when serving as a conduit for silt and crushed rock being flushed underground for filling mine voids. Terra cotta pipe (Fig. 2) has been used in boreholes, and lasts a long time if the glazed inner surface of the pipe remains intact. Its installation in a borehole is very difficult and is recommended only where it is certain that there will be no subsidence of the surrounding territory. Accessibility in order properly to join each section of the pipe is essential to a satisfactory job. In large water holes, where it is possible to lower a man into the hole for sealing the joints of the pipe, glazed terra cotta pipe has been used successfully for delivering mine water from pumps in the mines.

For large holes used for flushing refuse into mine workings where the surface wash is shallow, standard paving brick have been used to form a tube through the wash to the bedrock. These brick are laid radially, backed by common brick and cobbles laid in cement. It is obvious that the shaft through the wash must be large enough to permit a man to work in it. The flushing material is carried through bare rock from the juncture at bedrock to the openings underground. This plan has been used particularly where a cast-iron surface line is laid upon the surface to carry the flushing material from the breaker to the hole, and down the hole continuously, with no opening to the air at the top of the hole, thereby eliminating the necessity of an attendant at this point (Fig. 3).

Another similar type of borehole lining is constructed of hexagonal pieces of vitrified brick (Fig. 4). This type requires a steel drive pipe through the surface wash large enough to provide a finished tube of the desired size surrounded by a brick wall about 4 in. thick. The special bricks are laid in cement, six to a circle, around a lubricated pipe used as a form, and lowered in sections inside the large drive pipe; the bottom section is seated in a socket drilled in the solid rock, having the same inside

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diameter as the steel drive pipe. The remainder of the hole is drilled at the same diameter as the inside of the tube formed by the described segment-brick circle and is

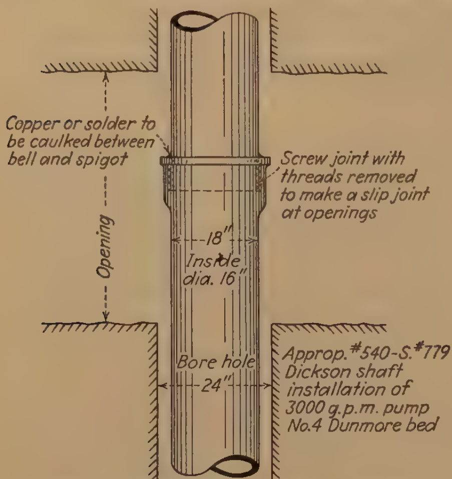

FIG. 1.—SCREW-JOINT CAST-IRON PIPE.

continued downward through the rock to the opening underground, where flushing is to be done. Generally, the finished flushing hole is about 8 in. in diameter and the inside of the drive pipe through the wash is about 20 in. The hexagonal brick are made up by the brick manufacturer, using special forms on special orders.

Such use of some form of clay products in lining has been adopted because no metal pipe has been found, within the limits of reasonable cost, that will withstand the abrasive effect of mine-refuse material flowing at high speed through it, and if the lining pipe cuts out in the hole, the hole may be lost, whereas metal flushing pipe underground in openings that are accessible is easily changed periodically.

Installation of Cast-iron Pipe

The installation of a cast-iron pipe for lining pump discharge holes through rock where, on account of seams, openings, or other conditions, it is not practicable to pump directly through the bare rock, is described below and illustrated by Fig. 1.

This pipe is specially cast, using a mixture of high-grade pig iron and steel scrap to provide a strong, tough casting, though sufficiently soft to permit the cutting of

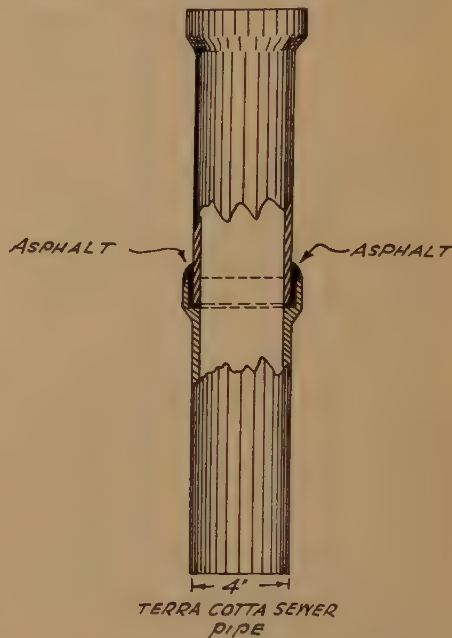

FIG. 2.—TERRA COTTA SEWER PIPE.

threads on the outside of the spigot end, and the inside of the bell end for screwing the sections together. Usually the pipe is cast in 10-ft. lengths to screw up at a net length of about 9 ft. 8 in. when connected. This pipe is placed in a large lathe and bored out to a square seat in the bell end, and both ends are squared up, so that when screwed together the sections will form an absolutely true line, with no kinks or bends. Such irregularities would prevent their insertion in a borehole, as the cast-iron pipe is absolutely inflexible. Some form of asphalt or heavy grease is used on the threads, as the pipes are up-ended over the holes and screwed together in that position. The threads being formed by templates are exactly standard, so that the pipe connection can be started by turning by hand if the unit is properly suspended over the topmost length of pipe in place in the hole.

Usually the pipe is dropped into the hole with the spigot end down; the clamps for holding the pipe in place grip it immediately under the bell as the weight is taken

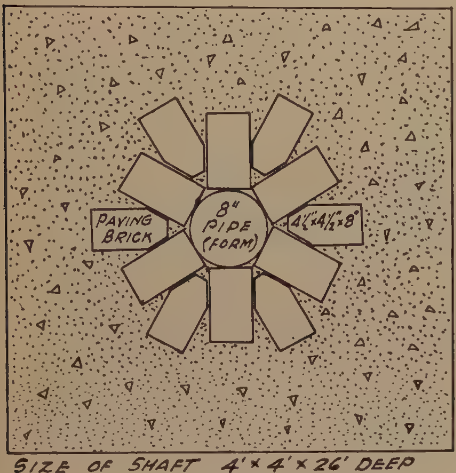

FIG. 3.—PAVING-BRICK LINING.

successively by the clamps and the crane described below.

Owing to the great weight of the cast-iron pipe—the shell being about one inch thick—only a limited number of sections can be supported for lowering into the borehole at one time, generally not more than 200 ft., or 20 sections. For lowering the pipe, sections are threaded over a standard shaft rope thrown over a sheave on a scaffold or gin pole erected over the top of the hole, while clamps upon this rope are operated by a heavy block and tackle connected to the drum of the drilling machine used in sinking the hole, or to a hoist of capacity equal to that of the drilling machine. The bottom of the first section generally is fitted with a heavy cast-iron block with a swivel eyebolt running through it, to which the cone of the lowering rope is connected. After the lining is finished, this block is removed from the bottom of the hole, working from the inside of the mine, by dropping it down and uncoupling the cone from the eyebolt, or unscrewing the eyebolt. This is not done until the lining has been cemented in place so that it cannot move after the rope and the surface clamps are disconnected. If the hole is deeper than approximately 200 ft., the pipe

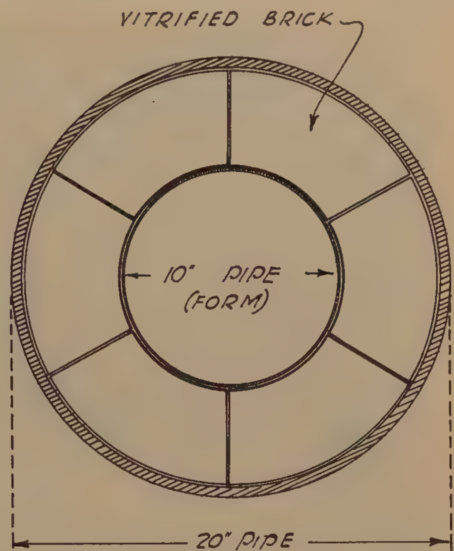

FIG. 4.—LINING OF HEXAGONAL PIECES OF VITRIFIED BRICK.

must be lowered in sections of about 10 lengths each, and instead of the sections being screwed together, the thread is ground off for a slip connection, or, if it happens that this connection can be made in an opening of an intermediate bed, the pipe can be fitted together there, working directly to screw the section together or by welding, or by pouring a lead joint.

This kind of lining has been used in many pump holes by The Hudson Coal Co. for more than 20 years, and has never failed by erosion or disintegration from acid. The space between this cast-iron pipe and the walls of the hole is poured with a thin mortar of neat cement, or of one part fine sharp sand to one part cement, which is allowed to settle and harden. Any settlement of the concrete is filled up by pouring more of the mixture until the pipe is absolutely secured in place by being completely surrounded with concrete, which runs into

the crevices, apertures and cracks in the sides of the hole through the bedrock.

Another method of installing cast-iron lining in pump holes is that used by the Glen Alden Coal Co. in a large hole at Truesdale colliery a few years ago. The lining was 24 in. inside diameter with a 1-in. shell, and was fitted together by ground slip joints. Because of the great weight of the pipe and the depth of the hole, which was about 671 ft. deep, the pipe units, about 10 ft. long, were lowered one at a time and were accompanied by a man riding on an improvised small carriage, which dropped down inside the pipe and was lowered with it, covered by a canopy. An escape trap was provided at the bottom of the carriage, to be used in case of accident. The rope connections were made to the pipe by means of lugs screwed fast to each side of the pipe by studs, which were unscrewed by the mechanic for the disconnection of the ropes after the pipe was slipped into place. After each length of pipe was placed, the space between the outside surface of the pipe and the hole wall was filled with mortar of about the consistency and ingredients described above, by pouring it around the pipe and tamping it with a slender rod until all space was filled to a point near the top of the pipe unit. The next unit of pipe was then similarly placed in position. Three ropes were used in the hole; two to handle the pipe and one to operate the man carriage. The single rope also operated the escapeway, so that the mechanic could be taken out of the hole by dropping through the trap door in the bottom of the carriage and thence down inside the pipe to the opening underground in case of an accident to the hole above the point at which the man was working.

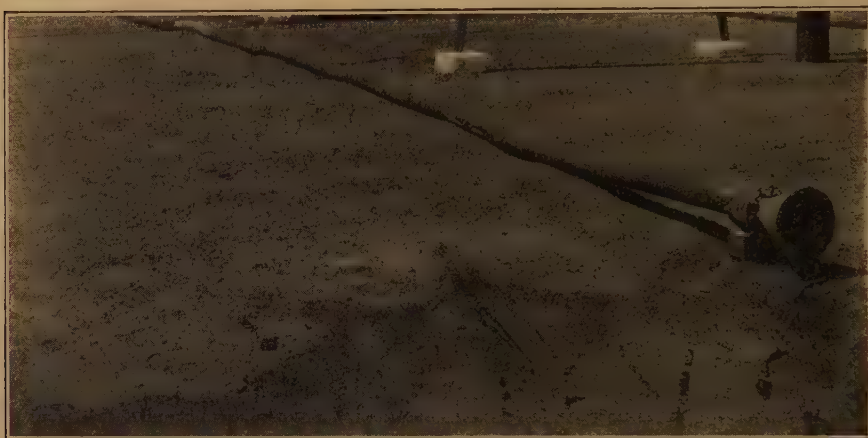

FIG. 5.—RUBBER PIPE WITH ACID-RESISTING BRONZE FUNNEL ATTACHED BEFORE INSERTION IN BOREHOLE.

The two ropes attached to the lugs on the pipe unit were needed to prevent the pipe from twisting and entangling the pipe and carriage ropes in the hole as the units were being lowered.

Among the principal reasons for this method of installing this kind of casing was the impracticability of handling a full string of the pipe screwed together, because of its great weight. Furthermore, by inserting one section at a time, greater assurance of complete cementing of the space around the pipe was obtained, because the mechanic could observe the pouring of the cement mixture around one section at a time. Also, mortar dropped in pails immediately after mixing and poured directly in small sections will not disintegrate and

separate to the same degree as it does sometimes when the mixture is poured from a great height, which is necessary when inserting large strings of pipe in a borehole and cementing after all the pipe sections are in place. In other words, the better cementing job is done when pouring behind one section of pipe at a time.

When the rock measures have not been disturbed by subsidence, and are reasonably dense, it is practicable to pump mine water through these holes directly to the surface with no lining whatever except through the wash and the last few feet at the lower end of the hole, where additional area is required for a foot joint. A great deal of pumping has been done and continues to be done through such holes in the northern Anthracite region. To accomplish the foot joint, the rock is carefully leveled and smoothed at the roof line and a thimble with a bolt flange is tightly wrapped in oakum above the flange and jacked up into the hole and forced into place. The foot elbow is then placed beneath it and jacked up, and the members are bolted together at the flanges. The elbow is supported by a concrete foundation. If necessary, a small hole is then drilled on an angle in the roof near the top of the thimble, and grout is pumped between the thimble and the bored rock tube to complete the connection, making it waterproof. Thus it resists the pressure created by the column of water standing and moving in the completed hole.

FIG. 6.—RUBBER PIPE BEING INSERTED IN BOREHOLE.

RUBBER PIPE FOR BOREHOLE LINING

The installation described in the following paragraphs and illustrated by Figs. 5 to 8 was made to eliminate the necessity of renewing every four months the steel wear pipe in the section of a borehole extending through the wash from the surface to the rock.

A flushing borehole was drilled during the early part of 1914 at the Marvine colliery of The Hudson Coal Co. A 12-in. dia. drive pipe was driven through the

wash to bedrock at a depth of 57 ft. below the surface. At a depth of 38 ft. quicksand was encountered and difficulty was experienced in cutting off the inflow of sand and water into the hole after the end of the drive pipe reached the solid rock. The hole was then continued with 12-in. diameter to a depth of 95 ft. from the surface, or 38 ft. into the rock, by drilling inside the 12-in. casing pipe, and driving the standard drive pipe through the hole in the rock in order to cut off the heavy pressure of water and quicksand. The hole for this distance was cased with 8-in. steel casing pipe, cemented in place, inside the 12-in. drive pipe, after which the hole was extended at 8-in. dia. to the bottom bed, an additional 475 ft., by drilling through the 8-in. casing pipe. A 6-in. steel lining or wear pipe supported at the top by clamps, and with the bottom end of the pipe free, was suspended inside the 8-in. pipe to protect the latter and the 12-in. drive pipe from wear.

FIG. 7.—ACID-RESISTING BRONZE FUNNEL ATTACHED TO RUBBER PIPE, WITH OAK CLAMPS, READY TO BE LOWERED INTO PERMANENT POSITION.

The character of the surface wash in the river basin, where the hole is located, is so variable that there was no assurance that a new hole could be successfully drilled through it to cut off the surface water at the junction of the wash and the top of the rock. Therefore, the original hole was kept open by cleaning it out from time to time when it became blocked, at considerable expense for both labor and material.

The wash material carries a high acid content and as a result the heavy 12-in. steel drive pipe and the steel 8-in. casing pipe inside it were eaten through from the outside at a depth of 66 ft. below the surface, and this action corroded the inner 6-in. casing steel pipe. It was thus affected by acid from both the inside and outside, as well as by the erosive effect of silt and rock-flushing material mixed with breaker wash water passing through the pipe into the mine workings.

During the year 1937, a drill crew spent several weeks cutting up and pulling out scraps of the 6-in. dia. pipe, which blocked the upper part of the hole. A 5-in. steel lining pipe was next resorted to because of incrustations within the hole through the rock of the unlined portion below the 95-ft. depth, which prevented the insertion of another 6-in. pipe. Two years later a drill crew was employed for about 3 months, chopping up and fishing scraps of this 5-in. steel pipe from the hole, because it had been eroded by abrasion and been eaten by acid water and hung in shreds in part of the hole.

To eliminate repetition of these fishing jobs, and to save repeated installation of 5-in. steel lining pipe, with the possible permanent loss of the hole, it was decided to experiment with a 4-in. rubber tube or pipe in one 90-ft. length, flanged on the upper end. The matter was given careful consideration by various rubber-pipe manufacturers, and finally the United States Rubber Co. agreed to construct such a tube of material approximately 1 in. thick.

This pipe is 4-in. inside diameter, 6-in. outside diameter, and 90-ft. over-all length. It is constructed of cotton duck and rubber, and the lower half is reinforced with helical wire. The inner lining of the tube is 3/8 in. thick and is composed of stock compounded to resist extreme abrasion. The outside cover of rubber is 1/16 in. thick. The carcass, or wall, of the pipe consists of plies of extra strong cotton duck 9/16 in. thick.

A specially constructed acid-resisting bronze funnel, with standard pipe-flange dimensions, provides for connection to the rubber pipe. It is supported under the flange with split oak clamps, bolted together. The rubber pipe weighs 647 pounds.

Special wire helicals reinforce the pipe for a distance of 3 ft. below the flange. There is no metal reinforcement for about 45 ft., but the remaining length has been reinforced with wire built into the carcass to protect the pipe from collapse if mud or silt should seep through a break that appears to be present in the 8-in. steel casing of the borehole.

Because the factory equipment limits the length to 50 ft., it was necessary to make the pipe in two lengths and splice them together. As a splice of this character had never been attempted before, it was necessary for the rubber company to submit the problem to its development department. After considerable experimentation and careful checking, a method of splicing was developed to construct a pipe that is uniform throughout—that is, the spliced section is the same as the remainder of the pipe. The splicing is considered an achievement by the rubber company, and, while at present the limitations as to maximum lengths are not known, it appears probable

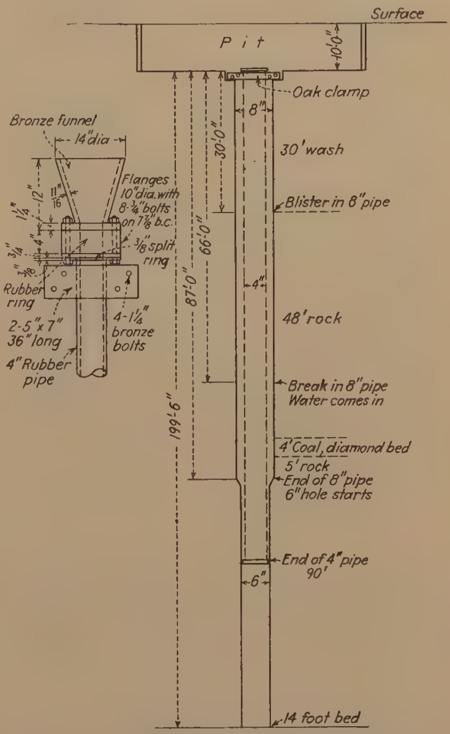

FIG. 8.—CROSS-SECTION OF BOREHOLE NO. 182-B, MARVINE COLLIERY, THE HUDSON COAL COMPANY.
Showing seams of wash, rock and coal penetrated by borehole.

that the use of rubber pipe has been considerably advanced for the purpose of lining boreholes.

The acid-resisting bronze funnel is used for the intake of material to be passed through the rubber pipe (Fig. 5). Receivers of this shape have a tendency to cause "swirling of the material," therefore it is advisable to use a 4-in. rubber ring to protect the flange, as well as the pipe itself. When the ring wears out, it is necessary to renew only the ring.

The 4-in. rubber pipe was installed on May 26, 1939, and six months after its installation it was examined by passing an electric light bulb through its entire length of 90 ft. The examination revealed the pipe to be in first-class condition. Similar examinations are to be made semiannually, to determine whether any wear has taken place.

Acknowledgment

The author desires to express his appreciation of the assistance and information furnished by Mr. E. W. Stahl, Assistant Mining Engineer of The Hudson Coal Co., in assembling this paper.

DISCUSSION

(H. H. Otto presiding)

H. H. Otto,* Scranton, Pa.—Throughout the Anthracite region of Pennsylvania, many boreholes of large diameters have been drilled for the express purpose of carrying power (electric, steam or air) from the surface into the mines. Pump holes for discharging water to the surface are numerous, and boreholes have been drilled for deposition of silt and waste material to be used for backfilling parts of the mines. There is a considerable amount of surface wash, varying from a few feet to 150 ft. over the rock. The bore hole must be properly cased through the wash and set on the rock before it can be drilled through the rock and coal seams. These conditions have caused trouble, particularly in the Wilkes-Barre and Scranton districts.

Mr. Johnson's paper touches briefly on some of the methods used to complete successfully not only the drilling but the preparation of the hole for use as a water, steam, or silt hole. Probably there are other holes in the region that have as difficult a problem as those described in this paper.

The use of rubber pipe as described is definitely a new departure in reducing the costs of maintaining a silt borehole where breaker silt and refuse are permitted to run back into the mines. The next logical step would be to find out to what extent or length a strong rubber pipe can be made and suspended from the top of a borehole without pulling apart at the top of the hole because of its own weight.

The difficulty at Marvine colliery was caused not only by acid water carrying silt down the borehole but also by acid water in the sand wash, which attacked the casing pipe from the outside at a point near the top of the rock.

* Mining Engineer, Hudson Coal Co.

Treated Mine Timber at Operations of Lehigh Navigation Coal Company Inc.

By Paul L. Burkhart*

(New York Meeting, February 1942)

Though at an earlier period brief studies had been made by the Lehigh Navigation Coal Company Inc., it was not until 1924 that J. B. Warriner, then general manager, called for a comprehensive study of timber treatment, treating solutions, and related operations. Because of the impossibility of procuring accurate costs and other pertinent treating data, the American Wood Preserving Co. agreed to treat a car of mine timber at the Port Reading plant of the Reading Railroad. Fig. 1 shows the treating data collected at the treating plant on this timber, which was installed in two of the Lehigh mines in 1925.

Table 1.—*Results of Timber Survey in 1925 before Treated Timber Was Introduced*

Length of tunnels and gangways, ft.	522,987[a]
Length of tunnels and gangways timbered, ft.	224,880
Number of steel mine sets in use	2,412
Number of timber sets in use	63,834
Number of timber sets with evidence of rot	50,419
Number of timber sets with evidence of squeeze	13,415
Percentage of all timber sets showing evidence of failure:	
By wet rot	6.0
By dry rot	73.0
By squeeze	21.0
Average life of wood set in Panther Valley, years	5.3

[a] 99.05 miles.

In the latter part of 1925, a survey of the tunnels and gangways of the company's Panther Valley mines was made to determine the proportion of gangways or tunnels timbered, cause of timber failures, approximate life of the average wood set and the approximate percentage of treated timber that it would be profitable for the company to use. Results of this survey appear in Table 1.

Because of the high percentage of timber that gave evidence of failure by rot, as noted in Table 1, and a further study of the life of the sections where timbering was necessary, it was deemed advisable to fill 25 per cent of the yearly gangway timber requirement with treated timber.

Quantities and Treatments of Treated Timber Purchased

In the early days of the program, none of the large treating companies would accept orders for small quantities of treated timber and the company was compelled to state the quantity of timber it would require at each operation in each month of the year. Table 2 is an example of the tabulation given to the various treating companies to assist them in making a yearly bid on the minimum quantity of timber required by the company during the year 1931. A bill of timber was calculated for each car, the weight of which would be between the minimum and maximum weight permitted for railroad-car loading. The treating company was paid for timber upon delivery at the operation specified.

As extra cars of timber were required, they were ordered from the treating company, and payment was required on their receipt, on the basis of the piece or tonnage

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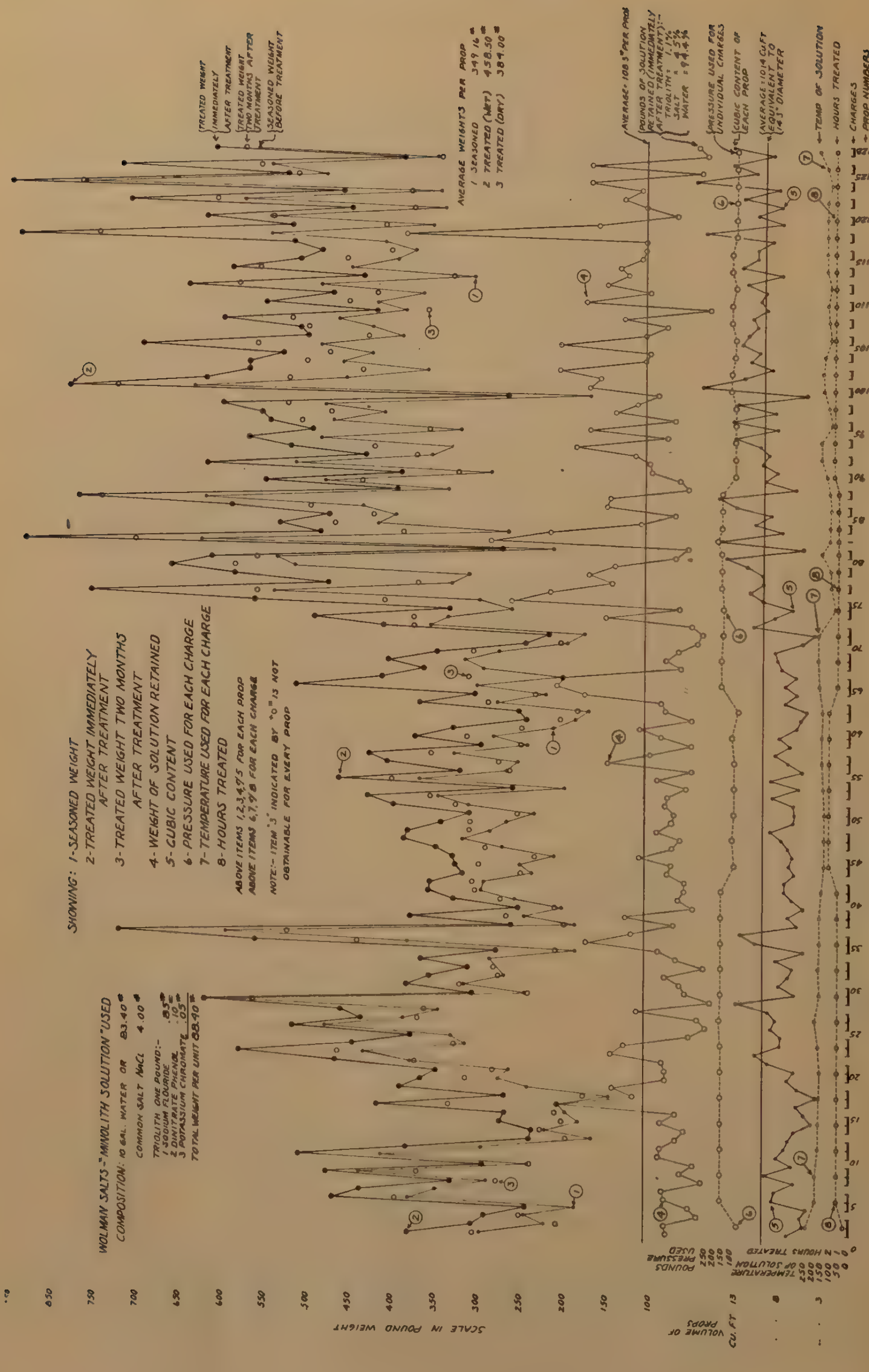

FIG. 1.—DATA PERTAINING TO GANGWAY TIMBER TREATED WITH WOLMAN SALTS.

rate that had been set for the minimum order.

During the years recorded, timbers treated in many different ways have been purchased and installed in the company's mines. The quantities of timber purchased, the treating processes and the solutions used are listed in Table 3.

Under "sets" in Table 3 are listed not the actual sets purchased but the sum of the lengths of all the timbers in those sets divided by the length of timbers in a single standard set, which consists of two legs, each 9 ft. long, and one collar 9 ft. long, or a total of 27 linear feet.

Peeled timber is either air or steam-seasoned according to specification; when the air-seasoned timber does not lose enough moisture to allow it to take the specified treatment timber is steam-seasoned for the necessary time to take treatment. "Green" denotes timber that has been cut less than 10 days, has lost only a small percentage of moisture, and has not had the bark removed.

Under "treating process" in Table 3 is given a brief description of the treating procedure. As a rule, air-seasoned timber is steamed for a short time, then a vacuum is applied for a length of time that will prepare the timber to take the specified treatment. Condensate is then drained from the cylinder, and then, without breaking the vacuum, the treating solution is run into the cylinder and pressure applied until a specified quantity of the solution has been injected into the timber. With full-cell treatment, the solution is drawn from cylinder without a final vacuum, which is applied only when the empty-cell process is specified. Steam-seasoned timber must be steamed longer and must be followed by a longer vacuum period and with a higher vacuum than the air-seasoned timber, or it will not take the specified treatment. The treating process "end impregnation, full cell," will be described later.

Solutions given are the percentage by weight of the salts in solution, and are as follows: Minolith; composed of ⅚ by weight of common salt and ⅙ by weight of Triolith (a proprietary salt); zinc chloride; zinc meta arsenite; and chromated zinc chloride; whereas the Aczol solution is a percentage by weight of the water-borne solution Aczol.

Under "retention per cubic foot" are recorded the pounds of dry salt or the gallons of a percentage of water-borne solution retained, with exception of timber treated by the end-impregnation process, which lists the quantity of salt solution per cubic foot collected on the free end of the sticks.

End-impregnation Process

The end-impregnation process of treating is similar to that patented by Dr. Boucherie in England in 1855, which injected a preservative solution into the ends of poles through a wooden head under pressure of 15 to 20 lb. per sq. in., using, however, copper sulphate as the preservative. A brief description of the present process follows:

1. Tree is cut down and trimmed not over 10 days prior to treatment.
2. Bark is not removed before or after treatment.
3. Logs are cut to length desired with ends cut off square.
4. At the end of the log, bark and about ¾ in. of wood is shaved off on a bevel to ensure that the rubber sealing gasket of the treating head will make a tight contact with the end of the log.
5. Log is then placed on a treating rack, and a metal treating head is placed on the end that has been beveled.
6. Treating solution containing a specified percentage of dry zinc chloride or chromated zinc chloride is then forced through the log under 25 to 50 lb. pressure.
7. Through the free end of the log, the treating solution under pressure pushes

out the natural sap and moisture in the cells.

8. The treatment is continued until the solution passing out of the free end of the log has a minimum concentration of 3 per cent zinc chloride or chromated zinc chloride.

9. Upon completion of treatment, the logs are removed from the treating rack, after which they are framed and are ready for shipment.

Fig. 2 shows the cutting tool that was first used for beveling the ends of the logs, and the tool cutting such a bevel. As workmen became experienced, this tool was no longer used; now the ends are shaved off with a draw knife or sharp ax.

Fig. 3 shows the cast steel treating heads and method of fastening these heads to ends of logs with steel rods; also sap and treating solution exuding from the free ends of the logs.

Table 2.—*Treated Mine Timber Required Year*

Month	Tamaqua Colliery							Greenwood Colliery						
	Collars or Bars				Range of Diameters, In.	Legs		Collars or Bars				Range of Diameters, In.	Legs	
	Number		Length					Number		Length				
	Pieces	Notch	Over-all	Notch to Notch		Pieces	Length	Pieces	Notch	Over-all	Notch to Notch		Pieces	Length
April	5 35	2 2	12′-0″ 9′-0″	10′-0″ 7′-6″	10–14 10–14	10 70	9′-0″ 9′-0″	34 6	2 2	9′-0″ 10′-6″	7′-6″ 9′-0″	14–16 14–16	68 12	9′-0″ 9′-0″
	One Carload							One Carload						
May														
June														
July	5 35	2 2	12′-0″ 9′-0″	10′-0″ 7′-6″	10–14 10–14	10 70	9′-0″ 9′-0″							
	One Carload													
Aug.														
Sept.								34 6	2 2	9′-0″ 10′-6″	7′-6″ 9′-0″	14–16 14–16	68 12	9′-0″ 9′-0″
								One Carload						
Oct.	5 35	2 2	12′-0″ 9′-0″	10′-0″ 7′-6″	10–14 10–14	10 70	9′-0″ 9′-0″							
	One Carload													
Nov.														
Dec.														
Equivalent 27 ft. Sets	121.68 Sets—3 Carloads							80.66 Sets—2 Carloads						

About ¼ min. elapses from the time the treating solution enters the treating head until the sap starts to ooze out from the free end of the log.

The wide range in the treating periods needed to complete this process is indicative of what occurs under the other pressure methods of treatment, and explains why some timber is treated so much better than other timber when in the treating cylinder for the same length of time.

A comparison of treating data of the end-impregnation process and the vacuum-pressure empty-cell process, which were taken at the treating plants, is shown in Table 4.

If high temperatures and pressures shorten the life of treated timber, which undoubtedly they do, the timber treated by end pressure should have longer life than timber otherwise treated, for the following reasons: Timber must be green to be thus treated properly (doty timber

1931 *by Various L.N.C. Co. Inc. Operations*

Coaldale Colliery							Nesquehoning Colliery							Total for All Operations		
Collars or Bars				Range of Diameters, In.	Legs		Collars or Bars				Range of Diameters, In.	Legs				
Number		Length					Number		Length							
Pieces	Notch	Over-all	Notch to Notch		Pieces	Length	Pieces	Notch	Over-all	Notch to Notch		Pieces	Length	Sets	Carloads	Pieces
25	2	8′-6″	7′-6″	10–12	50	7′-0″								131.35	3	415
			Props	8–10	100	8′-0″										
One Carload																
20	2	9′-0″	7′-6″	11–13	40	9′-0″								36.67	1	120
20	2	8′-6″	7′-0″	9–11	40	7′-0″										
One Carload																
							20	2	9′-0″	7′-6″	11–13	40	9′-0″	40.22	1	111
							7	2	12′-0″	10′-0″	11–13	14	9′-0″			
							9	1	13′-0″	12′-0″	11–13	9	9′-0″			
							6	1	14′-0″	13′-0″	11–13	6	9′-0″			
							One Carload									
20	2	9′-0″	7′-6″	11–13	40	9′-0″								77.23	2	240
20	2	8′-6″	7′-0″	9–11	40	7′-0″										
One Carload																
							20	2	9′-0″	7′-6″	11–13	40	9′-0″	40.22	1	111
							7	2	12′-0″	10′-0″	11–13	14	9′-0″			
							9	1	13′-0″	12′-0″	11–13	9	9′-0″			
							6	1	14′-0″	13′-0″	11–13	6	9′-0″			
							One Carload									
20	2	9′-0″	7′-6″	11–13	40	9′-0″	40	2	9′-0″	7′-6″	11–13	80	9′-0″	117.00	3	360
20	2	8′-6″	7′-0″	9–11	40	7′-0″										
One Carload							One Carload									
														40.56	1	120
20	2	9′-0″	7′-6″	11–13	40	9′-0″	20	2	9′-0″	7′-6″	11–13	40	9′-0″	76.89	2	231
							7	2	12′-0″	10′-0″	11–13	14	9′-0″			
							9	1	13′-0″	12′-0″	11–13	9	9′-0″			
20	2	8′-6″	7′-0″	9–11	40	7′-0″	6	1	14′-0″	13′-0″	11–13	6	9′-0″			
One Carload							One Carload									
							40	2	9′-0″	7′-6″	11–13	80	9′-0″	40.00	1	120
							One Carload									
197.14 Sets—5 Carloads							200.66 Sets—5 Carloads							600.14	15	1828

or wood that has been cut over one month cannot be treated successfully, as the pitch or sap coagulates preventing the treating solution from flowing freely through the log); no preliminary steaming is required; no preliminary vacuum is needed after steaming; period in which solution is injected into logs under pressure can be varied to suit variation in density of timber, etc.; preservative pressure and temperature are far less than with other methods, which would indicate less possibility of expanding and bursting the cells; and no final vacuum is provided, which would eliminate possibility that walls of wood cells would be collapsed.

FIG. 2.—CUTTING BEVEL ON END OF LOG WITH CUTTING TOOL.

FIG. 3.—TREATING HEADS, ALSO SAP EXUDING FROM FREE ENDS OF STICKS.

In following through tests of this treating process at the plant, it has been impossible to measure the quantity of solution entering the log, and the quantity noted in Table 3 under "retention per cubic foot" was the quantity of sap and treating solution collected on the free ends of logs. Some of the logs were cross-cut through the middle after treatment and 90 to 95 per cent of their cross section was found to be impregnated by the preservative.

As listed in Table 3, cost of treated timber varied from year to year, and reasons for some of this variation follow:

In 1925, at the Port Reading experimental plant of the Reading Railroad, one

TABLE 3.—*Treated Timber Purchased*

Sets	Seasoning	Treating Process	Treating Solution	Retention per Cu. Ft.	Cost per Ton
			Year, 1925		
42⅝	Steam	Vacuum, pressure, full cell	5.6 % Minolith	0.6 lb. salt	$32.50
			1926 and 1927		
205	Steam	Vacuum, pressure, full cell	3.0 % zinc chloride	0.479 lb. salt	24.08
78	Air	Vacuum, pressure, full cell	3.0 % zinc chloride	0.548 lb. salt	25.43
66	Air	Vacuum, pressure, full cell	2.5 % zinc chloride	0.547 lb. salt	25.43
77	Air	Vacuum, pressure, full cell	2.0 % zinc chloride	0.502 lb. salt	25.43
49	Air	Vacuum, pressure, full cell	6.0 % Aczol solution	3.056 gal. 6 % sol.	32.52
69	Air	Vacuum, pressure, full cell	5.0 % Aczol solution	2.538 gal. 5 % sol.	32.52
81	Air	Vacuum, pressure, full cell	4.0 % Aczol solution	2.76 gal. 4 % sol.	32.52
39	Steam	Vacuum, pressure, full cell	3.0 % zinc chloride	0.65 lb. salt	19.47
			1928 and 1929		
1457⅛	Air	Vacuum, pressure, full cell	3.0 % zinc chloride	0.76 lb. salt	19.47
1357⅔	Air	Vacuum, pressure, full cell	6.0 % Aczol solution	3.15 gal. 6 % sol.	29.23
149½	Air	Vacuum, pressure. empty cell	1.1 % zinc meta arsenite	0.234 lb. salt	19.93
			1930		
1380½	Steam	Vacuum, pressure, empty cell	1.2 % zinc meta arsenite	0.263 lb. salt	15.50
			1931		
669⅛	Steam	Vacuum, pressure, empty cell	1.2 % zinc meta arsenite	0.25 lb. salt	16.07
192½	Steam	Vacuum, pressure, full cell	3.0 % zinc chloride	0.81 lb. salt	16.19
			1932		
778¾	Air and Steam	Vacuum, pressure, full cell	3.15 % zinc chloride	0.81 lb. salt	11.20
			1933		
736⅛	Air	Vacuum, pressure, full cell	3.38 % zinc chloride	0.92 lb. salt	15.09
			1934		
733⅛	Steam	Vacuum, pressure, full cell	3.0 % zinc chloride	0.75 lb. salt	14.34
			1935		
243	Steam	Vacuum, pressure, empty cell	1.1 % zinc meta arsenite	0.25 lb. salt	10.81
352	Steam	Vacuum, pressure, full cell	3.0 % chr. zinc chloride	0.75 lb. salt	14.31
37¾	Green	End-impregnation, full cell	5.0 % zinc chloride	No record	9.00
			1936		
211	Steam	Vacuum, pressure, full cell	3.0 % chr. zinc chloride	0.75 lb. salt	13.97
274⅛	Green	End-impregnation, full cell	5.0 % zinc chloride	3.56 gal. 5 % sol.	12.00
			1937		
893	Green	End-impregnation, full cell	5.0 % zinc chloride	3.48 gal. 5 % sol.	10.50
			1938		
365¾	Green	End-impregnation, full cell	4.0 % chr. zinc chloride	4.0 gal. 4 % sol.	11.50
			1939		
845	Green	End-impregnation, full cell	4.0 % chr. zinc chloride	3.63 gal. 4 % sol.	12.00
			1940		
899½	Green	End-impregnation, full cell	4.8 % chr. zinc chloride	3.40 gal. 4.8 % sol.	13.00

car of timber was treated with Minolith, a proprietary Wolman salt. An excess of labor was required at the plant while treating and procuring treatment data. The salt cost $0.085 per pound, which made the cost $1.00 per ton more than if zinc chloride had been used, but in this calculation no allowance is made for the value of the salt in solution, which was wasted after treatment of this timber was completed.

TABLE 4.—*Comparison of Data on Treating Processes*

		Vacuum Pressure, Empty-cell Process	End-impregnation Process
Steam	Period, hr.	8	None
	Maximum temperature, deg. F.	235	None
	Maximum pressure, lb.	20	None
Vacuum	Period, hr.	2	None
	Maximum pressure, min.	25	None
	Minimum temperature, deg. F.	130	None
Treating	Period, hr.	3¼	¾ to 2½
	Maximum pressure, lb.	175	50
	Average temperature, deg. F.	120	90
Vacuum	Period, hr.	½	None
	Maximum pressure, min.	25	Slight
	Minimum temperature, deg. F.	140	None

Between 1926 and 1929, no competitive bids were received for treated timber, and the company receiving the contract during these years insisted that timber would not take the specified treatment unless it was air-seasoned previous to treatment. The timber was insured while seasoning, and cost of insurance was included in the cost of treated timber.

Aczol-treated timber costs more than timber treated with zinc chloride. Aczol, a proprietary preservative solution, costs $1.06 per gallon, and this cost accounted for $5.00 difference in cost per ton of timber between the two preservatives, without considering, however, the value of the Aczol solution wasted upon completion of treatment. The contract for treated timber in 1926 called for treatment of 625 sets of timber, which were delivered by the middle of 1927, whereupon a contract was made with the same company to treat 1500 sets of timber with zinc chloride and 1500 sets with Aczol. A number of anthracite companies placed orders for treated timber with the treating company at this time, which was reflected in lower prices for treated timber.

From 1930 to 1936, tables similar to Table 2 were sent to the various treating companies in each year for bids on a year's minimum quantity of treated timber. Bids on the 1930 contract were received from six companies in the latter part of 1929, and the bid of the company receiving contract was $3.46 per set lower than the bid of the company holding contract the preceding year.

The 1932 contract was given to a new treating company, which to get started was willing to cut profits. The peeled timber was purchased direct by the treating company from timber suppliers, therefore middleman's profit was eliminated.

The 1933 contract was given to the company submitting the lowest bid. This company had the contract for 1932, and the price for 1933 was in line with price paid prior to 1932.

The company receiving contract for the Z.M.A.-treated timber in 1935 had on hand at the plant rejected poles, which could be converted into mine timber. It was possible for them to sell this treated timber at a low price and clean up this excess. This same year, the first car of end-impregnated treated timber was received from a small treating company, which was satisfied to recover the cost of untreated timber while developing the technique of treating by this method.

From the latter half of 1936 to date, the end-impregnated treated timber has been purchased from small companies having plants close to the woods. These companies purchase timber direct from timber-

men, and use local labor at the plants. The minimum labor rate according to the Fair Labor Standards Act is paid. Since 1937, increases in cost of treated timber are due to increased timber cost, labor and freight rates.

Inspection of Timber

Upon arrival at wharf, timber is checked against order as to sizes, manufacture and quality, after which a representative number of timber borings are taken and tested to check depth of treatment. Also, several times during the year treating plants are visited to follow through on treatments, as it is possible at the wharf to determine the depth of treatment but not the percentage of treating solution used, which governs, to a large extent, the life of treated timber. Any undesirable timber or timber that is not treated according to specification is rejected, and a corresponding reduction is made in the amount of the bill for the rejected timber.

Placing and Tagging of Timber

For comparative record of life, sticks of treated and untreated timber were tagged and placed in sections where at the time of placement it was thought that failure would be caused by dry rot.

Between the years 1925 and 1931, in different sections of the various mines in the Panther Valley, 8364 pieces of treated and 2093 pieces of untreated timber were tagged and placed in service. A record was compiled of the date and location of placement of these sticks according to tag numbers and treatments. At yearly intervals, since treated timber was first tagged and installed, inspections of the tagged timber have been made; and cause of failures or reasons for timber removal have been noted and recorded, as also the condition of the timber remaining in service.

Service Life of Tagged Timber

A comparison of the percentage of tagged pieces that fail and the average life of these failures are listed in Table 5. Table 6 lists the average service life of the various treatments of timber by causes.

Table 5.—*Comparative Service Life of Tagged Timber*
September 1940

Cause of Failure	Percentage Failed and Life	Treated	Untreated
Dry Rot	Percentage of tagged pieces failed	8.2	58.8
	Average months service life	107	68
Squeeze	Percentage of tagged pieces failed	19.7	18.9
	Average months service life	59	36
Robbed Out	Percentage of tagged pieces failed	25.3	11.7
	Average months service life	76	59
Sealed Off	Percentage of tagged pieces failed	0.5	0.6
	Average months service life	22	24
Other Causes	Percentage of tagged pieces failed	4.5	6.6
	Average months service life	57	28
All causes	Percentage of tagged pieces failed	58.2	96.6

"Dry rot" failures are caused by fungus growth, impairing the strength of timber, whereas "Squeeze" failures are caused by top or side pressure, which breaks timber. "Robbed out" timber designates timber that was placed in sections that were abandoned while the timber was still serviceable, and service life covers the time between placement of timber and abandonment of section. Timber classed as failing from "other causes" was timber knocked down by trip derailments, removed for erection of chute spouts, and so forth; the identity of the timber was lost after it had given the length of service noted.

Condition of Tagged Timber Remaining in Service

The comparative condition and months of service life to date (September 1940) of

the tagged timber remaining in service is listed in Table 7, and Table 8 lists the service life to date (September 1940) according to condition of the various treatments of tagged treated timber remaining in service. In these tables condition of timber is noted as "good" when it shows cracks extending from surface to neutral axis and noticeable bending in the stick with no transverse cracks appearing on the surface, and "bad" such timber as has large longitudinal cracks and transverse cracks with obvious bending, which will have to be replaced within three months.

TABLE 6.—*Service Life of Tagged Treated Timber*
SEPTEMBER 1940

Treatment	Causes of Failure: Dry Rot	Squeeze	Robbed Out	Sealed Off	Other Causes
	Months Service Life				
6 % Aczol air-seasoned	102	52	77	24	63
4 % Aczol air-seasoned	98	65			56
3 % zinc chloride air-seasoned	109	61	67	20	52
3 % zinc chloride steam-seasoned	134	76	90		80
2½ % zinc chloride air-seasoned	109	7			116
2 % zinc chloride air-seasoned	110	66			62
Zinc meta arsenite ¼ lb. steam-seasoned	87	66	37		
3 % sodium fluoride, open tank	97	76			24
Wolman salts steam-seasoned	92	84	91		
Weighted average all treatments	107	59	76	22	57

TABLE 7.—*Comparative Condition and Life to Date of Tagged Timber Remaining in Service*
SEPTEMBER 1940

Defect	Condition of Timber	Treated: Per Cent	Treated: Months Life	Untreated: Per Cent	Untreated: Months Life
None	Good	22.1	136	0.6	146
Dry rot	Slight	11.1	138	0.5	138
	Fair	4.5	143	1.7	141
	Bad	0.5	150	0.6	131
Squeeze	Slight	1.3	139		
	Fair	1.8	140		
	Bad	0.5	139		
Total remaining in service		41.8		3.4	

no signs of rot or squeeze. Under the group head "dry rot," "slight" designates timber in which rot is not over ½ in. deep, "fair" when it is not over 1½ in. deep and "bad" when it is so far rotted that it will need to be replaced within six months. Under the group head "squeeze," "slight" includes timber in which small longitudinal cracks below the neutral axis have occurred with little bending of the stick, "fair" includes timber with large longitudinal

VALUE OF TREATED TIMBER

At the present time (September 1940), the company pays $7.75 per ton for untreated and $13.00 per ton for treated timber. The costs shown in Table 9 are based on these prices.

In compiling Table 9, average life to date (September 1940) of treated and untreated timber that failed was used, and it is realized that the ultimate saving, when

all the treated timber has been classed as failed, will be far greater than that shown in the table.

The zinc chloride treated (end-impregnated) tagged timber, in service 72 months as of November 1941 is in good condition, whereas untreated tagged timber placed at solutions used, and whenever a new method or treatment has appeared better or more economical than the one in use, the better and cheaper treatment has been introduced. Timber end-impregnated with chromated zinc chloride is the company's present standard.

TABLE 8.—*Service Life (Months) of Tagged Treated Timber Remaining in Service* SEPTEMBER 1940

Treatment	Good Condition	Dry Rot			Squeeze		
		Slight	Fair	Bad	Slight	Fair	Bad
6 % Aczol air-seasoned	138	136	139	142	134	138	136
4 % Aczol air-seasoned			157				
3 % zinc chloride air-seasoned	149	154	153	162		150	156
3 % zinc chloride steam-seasoned	142	142	142	144	148	149	136
2½ % zinc chloride air-seasoned			158				
2 % zinc chloride air-seasoned	153	156	157		155	155	156
Z.M.A. ¼ lb. steam-seasoned	115	118	139		122	110	117
Wolman salts, air-seasoned		176	176	176			176
4 % zinc chloride end-imp., full cell	72[a]						

[a] Inspected November 1941.

TABLE 9.—*Comparative Costs of Treated and Untreated Timber*

	Failure			
	Dry Rot		Squeeze	
	Treated	Untreated	Treated	Untreated
Cost of timber per set	$10.88	$ 6.49	$10.88	$ 6.49
Cost of labor placing, per set	14.80	14.80	22.34	22.34
Cost of timber in place per set	$25.68	$21.29	$33.22	$28.83
Average life of timber, years	8.92	5.67	4.92	3.0
Ratio of service	1.00	1.57	1.00	1.64
Cost per life of treated set	$25.68	$33.43	$33.22	$47.28
Savings by use of treated timber	$ 7.75		$14.06	
Percentage saved per set on additional expenditure of $4.39 for treated timber	23.2		29.9	

the same time had to be removed after 54 months service, because of dry rot. It is expected that the end-impregnated timber will give as good if not better service than the other timber treatments, and savings will be greater than those noted in Table 9 under the major head "failures."

SUMMARY

Much time and study have been given to timber treatment, methods and treating

It is important when dealing with timber preservation to make a distinction between underground and surface requirements. The company's experience with underground timber failures due to ground pressures, which cause longitudinal checking, indicate that the method of treatment should be the first consideration. Any method that gives a desired heavy concentration of preservative throughout the whole stick should prove beneficial with most of the approved preservatives.

DISCUSSION

(*D. Clinton Helms presiding*)

H. H. OTTO,* Scranton, Pa.—There is no question that treated timber generally lasts longer than untreated. However, the approximate life of a section of the mine workings in which timber is to be used must be known. The life of the timber should be spent when the use of the mine opening has been completed.

Generally, it does not pay to use treated timber of any kind in the northern anthracite field, except on main transportation roads or airways. In the past 10 or 15 years a great deal of timber has been trucked into the northern territory. This timber usually is cheap and frequently it is more economical to retimber than to put the extra money in treated timber, especially for ties underground.

There is no question that the use of treated ties pays around the foot of shafts that have a long life, and in standard-gauge yards of long life. Treated timber should be used whenever desirable, but only when the life of the mine or section of the mine justifies it.

P. L. BURKHART (author's reply).—We have found that wherever untreated timber fails by dry rot before a section is abandoned, it pays to place treated timber in any part of the section in which replacements of untreated timber would be necessary during the life of the section.

* Mining Engineer, Hudson Coal Co.

R. D. HALL,* New York, N. Y.—Were the locations, both of treated and untreated timber, so chosen as to give a representative indication of their resistance to deterioration?

P. L. BURKHART.—All gangways, tunnels and airways were dry when tagged timber was placed in them for test, and it was expected that the primary cause of failure would be dry rot. As time went on, some of these openings became wet, as the coal measures above them were worked out.

In sections where timber is continuously wet, the presence of water favors the untreated timber by preventing dry rot, and treatment of timber is of no advantage because the water leaches out the water-borne preservative. In Table 7 the majority of the tagged untreated timber remaining in service is in sections where the timber is continuously wet.

Where timber is alternately wet and dry, the untreated timber rots rapidly whereas the treated timber is not affected by rot until the majority of the preservative material has leaked out.

* Engineering Editor, *Coal Age*.

Shuttle-car Haulage in West Virginia

By D. L. McElroy,* Member, and John L. Schroder, Jr.,† Student Associate A.I.M.E.

(New York Meeting, February 1941)

Although the earliest use of rubber-tired haulage was in Illinois in 1936, the first unit of this type of equipment used in West Virginia was shipped into the state in 1938. All units placed in West Virginia mines have been of the shuttle-car type. The Katherine Coal Mining Co., at Lumberport, W. Va., was the second company in the United States to install shuttle cars and the Gay Coal and Coke Co., at Mt. Gay, W. Va., was the first company in the country to install the thin-bed type of shuttle car. On Oct. 8, 1940, nine companies in the state were using 41 shuttle cars, of which 10 cars at three mines were of the thin-bed type. All shuttle cars of the low type were in the southern fields and all of the high type were in northern West Virginia. At that time several units were in the process of delivery.

The data in this paper were collected in the field by Mr. Schroder in connection with graduate work on mechanical loading of coal in West Virginia by mobile loaders. In the course of his field work he visited five mines in the state that were using shuttle cars. All of these mines were in high-volatile coal beds of strong structure.

Table 1.—*Data on Loading Machines and Shuttle Cars*

Per Shift Basis

Number of Mine	Thickness of Bed, In.	Maximum Distance of Shuttle-car Travel, Ft.	Number of Shuttle Cars Serving Machine	Number of Men in Crew[a]	Number of Places per Machine	Width of Places, Ft.	Capacity of Shuttle Cars, Tons	Capacity of Mine Cars, Tons	Tonnage per Loading Unit	Tonnage per Shuttle Car	Tonnage per Man on Crew	Tonnage per Loading Unit per Foot of Bed Thickness	Tonnage per Loading Unit per Cubic Foot of Cut
1	72	700	3	16	14	12	3	6	513	171	32.10	85.5	1.19
2	78	720	3	18	24	14	4.8	4	475	158	26.40	73.1	0.65
3	90	700	3	21	10	14	4.5	3.1	500	167	23.80	66.7	0.60
4	90	500	2	16	12	12	4.4	2	350	175	21.90	46.7	0.47
5	46	500	2	18	10	30	3	{2.2[b] / 1.6}	300	150	16.66	78.3	0.44

Note.—All blasting done with permissible explosives.

[a] Includes all men required to supervise, prepare, load and haul coal from face to mine cars at transfer station.

[b] Approximately 50 per cent of cars have 2.2 tons capacity.

The more pertinent data obtained at the mines visited are given in Table 1.

The maximum distance of shuttle-car haul is the distance between the transfer station and the farthermost place. Tonnage per loading unit per foot of bed thickness is to some degree a measure of relative efficiency, especially if the different operations are working the same bed or have somewhat similar conditions. The thin-bed operation is well up in the higher bracket of the afore-mentioned column, which is as

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it should be. In this mine the entries are driven very wide and the roof gives little trouble, so that conditions are especially favorable for shuttle-car operation. It will also be noticed that the three mines using three shuttle cars to serve the machine are averaging about 500 tons. The use of a third car is not the entire reason for this larger tonnage but probably is the greatest contributing factor.

A definite relation exists between the capacity of the mine car and the shuttle car, which may not be apparent. When the shuttle car is forced to stop and start its conveyor while unloading, peaks in power consumption are increased and time is lost. If a small mine car is in use, the elevating conveyor at the transfer station must be stopped at intervals if the mine cars are to be loaded properly without undue spillage, or must use spill plates. Some operators are advocating larger mine cars, so that more than one shuttle-car load will be needed to fill them. One mine is using 6-ton mine cars, which will hold the coal from two shuttle cars (Table 1).

The tonnage per loading unit per cubic foot of cut introduces the natural and mechanical factors as well as the efficiency of the operation. This is governed by the width of the working places, the length of cutter bar, the thickness of bed and the tonnage per loading unit. It is interesting to note that the mines using three shuttle cars to serve the machine haul as much coal per unit as those using two cars. This further shows the advantage of the third car, for, according to the table, an increase in tonnage is obtained almost equal to the tonnage one car hauls in an ordinary two-car system. Tons per man on a crew is likewise increased where the third shuttle car is used.

Mining Methods

In none of the operations visited in West Virginia are shuttle cars being used with the conventional room-and-pillar system. The block system is used exclusively and the size and shape of the blocks depend upon the character of the top and bottom as well as on the method of pillaring. If the entries and breakthroughs can be driven 14 ft. wide, without excessive timbering, shuttle cars can travel around a 90° turn, but if driven less than 14 ft. wide the breakthroughs should be driven at less than a right angle. One operation in northern West Virginia has experimented with the angle on which the breakthroughs should be turned and found a 60° angle to be sufficient, although a 45° was used originally. Another alternative in narrow places is to make the first cut on a 45° angle and then swing back to the 90° angle on the second cut, if it is desired to maintain square blocks.

A novel system is to follow development work with pillar work, so as to have a common transfer station somewhere near the center of production. The work is thus greatly concentrated, which is of prime importance in any form of mechanical loading. There is the further advantage, that cleaning up of entries is avoided in the pillar work, since the blocks are mined soon after development. Furthermore, about twice as much coal goes over each transfer station, which saves time and reduces the cost of moving the transfer stations.

An important factor in laying out the system of mining is the distance the places are allowed to advance from the last transfer station, or the distance between the stations. What these distances should be is largely in controversy at present, as distances from 400 ft. to 1200 ft. have been advocated. When a third shuttle car is used, the distance can be somewhat lengthened, consequently a greater tonnage goes through each transfer station. The ultimate aim is to get the greatest tonnage possible through each transfer station without increasing the delay at the loading machine by waiting for shuttle cars.

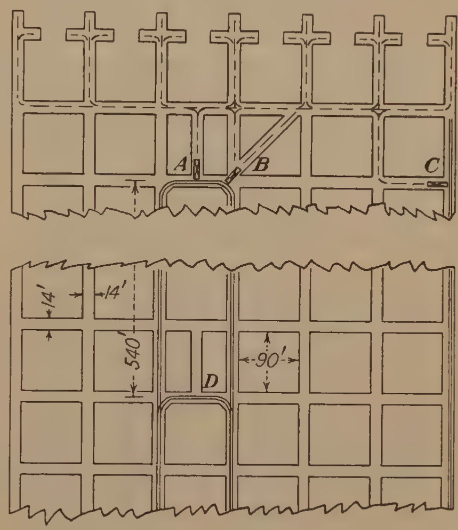
In the opinion of most men familiar with shuttle-car operation, the third car behind the machine more than pays for itself. With the third car the distance between transfer stations can be increased and if one of the cars should break down production is not cut approximately in half, as happens when only two cars are in operation.

Breakdowns naturally accompany any mechanical system, therefore it is considered good practice to have the necessary repair parts on the section, so that production is not seriously reduced when a shuttle car is out of service. Especially is this true when but two cars serve a loading machine. Some of the necessary parts include: two tires and a conveyor motor for each shuttle car, and two tram motors for each shuttle-car section. Time is saved by replacing a damaged motor with a spare one instead of dismantling the damaged motor or taking it outside to be repaired while the car stands idle.

Six to 11 entries or rooms are driven with one loading machine; the majority of the operations drive 6 or 7 entries. When 6 or 7 entries are driven, a machine has from 10 to 16 working places, including breakthroughs. Experience indicates that 5 entries do not furnish enough places to maintain a smooth cycle of operation for a shuttle-car system.

Shuttle-car Transfer Stations

The positions of transfer stations vary, as shown in Fig. 1, being at the center of the group of entries (at *A* or *B*) or at the outside of the group (at *C*). When the transfer station is in or just off the center of the group, the haulage is usually in the form of a run-around with the station either in a split of a block or in a regular entry. A grade running at right angles to the direction of the entries usually is the principal reason for placing the transfer station in the outside entry, so that all the loaded shuttle cars move downgrade. One of the operations visited used a spare unit of transfer-station equipment for the purpose of minimizing the lost time between changes of location of transfer stations. The extra equipment is set up in the proper place and



KEY
═ Track
--- Shuttle-car routes
▬ Transfer stations

Fig. 1.—Location of underground transfer stations.

the move to this discharge point is not made until everything is complete, at which time the coal begins to go over the new transfer station without any loss of time. Other operations make their moves of transfer stations on off shifts so that no productive time is lost on this account. Ordinarily the transfer stations can be moved and set up ready for operation in 3 to 6 hours.

Coal can be transferred from the shuttle cars to the mine cars or conveyors in many ways. All the operations in West Virginia employ elevating conveyors at the transfer stations, shooting both top and bottom to install them. This conveyor is an ordinary chain-flight elevating conveyor powered by a 7½-hp. motor and mounted on mine-car wheels, so that it can be moved on the track. The elevating conveyor may be eliminated

by lowering the track, thus allowing the shuttle cars to dump directly into the mine cars from a platform. Still another system has the shuttle cars dumping directly onto conveyors. Neither of the last two methods is used in West Virginia. With elevating conveyors, a ramp is used, or the bottom is shot out at the point where the shuttle cars discharge their load. It is much more desirable to take up bottom because shuttle cars then do not have to tram up a ramp, which causes peaks in power consumption. If the storage capacity of the elevator is inadequate and the mine cars are too small, the shuttle-car conveyor must be started several times while discharging into mine cars. With mine cars of large capacity, this problem is negligible. Sideboards built around the bottom of the elevator usually provide sufficient storage capacity so that power peaks on the batteries are reduced.

In general, mine cars are pulled under the elevator of the transfer station either by a hoist controlled at the loading point or by a locomotive. Operators using locomotives claim advantages for them, which are strengthened by the fact that the locomotives were on hand when the shuttle cars were installed. The locomotive is flexible in that it can push the cars back if they advance too far. Offsetting this advantage, however, the hoist causes less delay throughout the shift, since it is controlled directly at the loading point; therefore, no time is lost relaying signals. A disadvantage of the hoist is that it must be moved and set up at each transfer station. Then too, the locomotive is already connected to the trip, so that the locomotive pulls the loaded trip without delay.

Conditions Affecting Operation

Timbering resolves itself into setting crossbars because the shuttle cars and machines working across the face require the full width of the place, leaving no room for posts. Crossbars are used in both systems of haulage. The cost of timber is virtually the same as for the track system, since the crossbars are also used. The Department of Mines requires a definite number of crossbars over the loading machine, depending on the coal bed and the district. This timbering increases the work of the shuttle cars, in that they must bring the timber to the faces. Thus, one of the most important items in the efficient operation of shuttle cars is the character of the top. If the top requires close timbering and narrow entries, it is very difficult for the shuttle car to negotiate turns without backing up once or twice, or to tram with any degree of mobility. Then, too, it creates hazards for the shuttle-car operator, as he may be squeezed against the timber or the timber may be knocked out.

In Table 1, mines 1 to 4 inclusive have a roof of about one foot of roof coal overlain by draw slate. Mine 5 has a strong slate roof.

Character of the bottom is equal in importance to the top conditions. As long as the bed is without adverse grades the character of the bottom does not seem to be a major obstacle, although with a soft bottom tonnage will be reduced and probably spare battery units will be required. In wet sections the muck is as much as 6 in. thick in places and yet the cars continue to tram, but with delay and an increase in power consumption. In some operations, especially those with wet bottoms or excessive grades, batteries are changed once during a shift to keep the shuttle cars from slowing down. Usually the batteries are changed without loss of time, if the battery change is coordinated with the maneuvering of the loading machine. To overcome the disadvantage of mucky bottoms, "corduroy" roads are made by laying down waste timber slabs or planks. Muddy roads are sometimes made passable by having the loading machine clean the shuttle-car haulways as needed. Some companies are looking for a pump that will handle the muck.

Time Study

The time study, as summarized in Table 2, was taken by the co-author on a form that is in general use by some companies using shuttle cars. Fig. 2 shows graphically time-study data on a loading machine and a shuttle car for one working place. Operations for the whole shift can be plotted in this manner. As shown in Fig. 3, the entire work was in a pillar section.

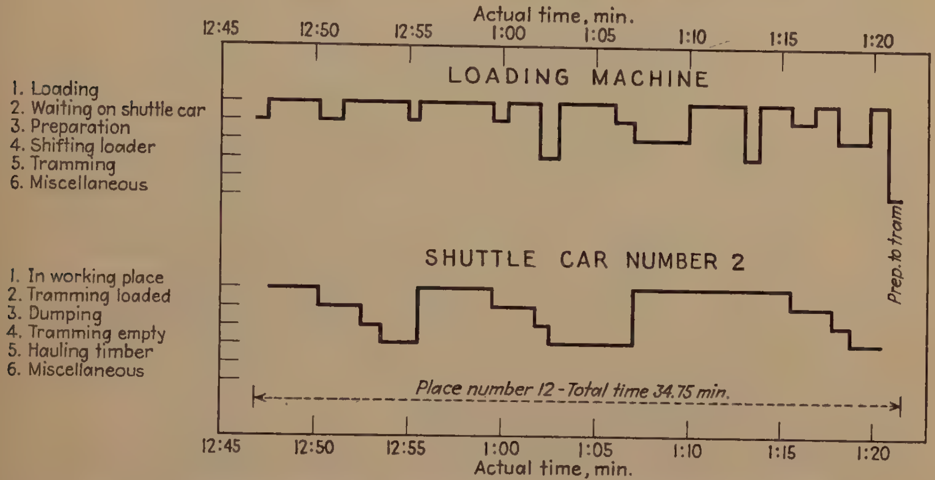


Fig. 2.—Graphic time-study chart, shuttle-car section, Oct. 10, 1940.

The numbers adjacent to the working places in Fig. 3 indicate the sequence in which the places were worked. Places showing two numbers were loaded out twice during the shift. This was the first time study taken on 100 per cent pillar work at this mine and, considering the conditions, the performance compared favorably with previous time studies. This study was made after the mine had been idle several days

Table 2.—*Summary Time Study*[a]

Mine ____________________ Date October 29, 1940

Equipment: One loading machine and two shuttle cars.

Operation	Minutes	Per Cent
Total time loading coal	203	44.7
Total time loading slate	4¼	0.9
Shifting loader in working place	21¼	4.7
Repairs to loading machine	0	0.0
Tramming loading machine	55½	12.2
Preparing coal	46½	10.2
Roof condition and timbering	½	0.1
Power	0	0.0
Repairs to shuttle car	0	0.0
Waiting on shuttle car	104¾	23.1
Shifting shuttle car	9¼	2.0
Lubrication of loading machine	1½	0.3
Preparing to tram loading machine	8	1.8
Total time	454½	100.0

Number places cleaned up, 16. Avg. time for each place, 24.3 min.
Number feet loader trammed, 3290 ft. Avg. feet trammed per min. 59.3 ft. per min.
Number of shuttle cars loaded, coal, 70; slate, 2; total 72
Number mine cars loaded, coal, 158; slate, 2; total 160
Estimated tons coal loaded this shift, 332

[a] During the shift the loader had to wait a total of 10 min. for preparation before entering; i.e., had to wait until the shotfirer finished shooting the place.

preceding the day of the study, therefore conditions were particularly bad, since on pillar work interruptions permit the roof to settle. Especially was it disadvantageous in these places because some kerfs had practically closed, making it difficult to obtain satisfactory blasting. This accounted in part for the time lost because of improper preparation. At this operation a top cutting machine, mounted on caterpillars, was used.

This time study (Table 2), in comparison with several others, represents closely a typical performance but is better than average for pillar work. This and several other time studies taken indicate that, other operations remaining the same, the reduction of the waiting time on shuttle cars is the most logical operation in which to save time and improve performance. One of the main purposes of the shuttle-car operation is to reduce the shifting time at the loading machine as compared to loading directly into mine cars. It should be kept in mind that the advantage of shuttle cars over mine cars in shifting time is reduced in proportion to the increase in the capacity of the mine cars. When a mine car having the same capacity as that of the shuttle car is used, a proper load balance must be maintained between the two. Since a part of a shuttle-car load would partly fill a mine car, which must go to the tipple or throw the loading cycle out of balance, the loading cycle should be arranged so that the mine cars are advanced when no shuttle car is at the station. It is claimed that more advantages are gained with the use of a mine car larger than the shuttle car

FIG. 3.—SECTION ON WHICH TIME STUDY WAS TAKEN.

because the elevator can load more between changes of the mine cars.

The three main nonproductive items of the study in order of their time consumption are: (1) waiting on shuttle cars, 23.1 per cent; (2) tramming the loading machine, 12.2 per cent; and (3) preparation of coal for the loading machine, 10.2 per cent. Time of waiting for shuttle cars probably offers the best opportunity for an increase in efficiency; because after proper study of road conditions, distance of working places from the transfer station, and careful selection of traveling routes the interval between arrivals of shuttle cars can be timed to be equal to or less than that required for the shifting of cars at the loading machine. With an average car-change distance of 50 ft., the data indicate that total working time consumed in waiting on shuttle cars can be reduced to approximately 12 to 15 per cent. In other words, if the percentage exceeds that figure it probably is due to lack of organization or the inefficiency of one of the aforementioned items. Although tramming the loading machine and preparation of coal for the loading machine are not directly related to shuttle-car performance, the whole cycle of production is centered around the loading machine. In this particular study, the time of tramming the loading machine was 12.2 per cent of the total time. For all practical purposes the only way to reduce this is by maximum concentration of working places and increase in the tramming speed of the loading machine. Studies made in West Virginia mines indicate that material improvement can be made along these lines. The percentage of time used in preparing places was 10.2 per cent. Other studies at this particular mine showed this delay to be between 8 and 10 per cent. This factor, however, resolves itself strictly into a problem of organization and supervision, and it is believed that under the best of organization and management it can be reduced practically to zero. Studies made at other mines bear out the preceding statement.

The only other item amounting to over 2 per cent of the time is that consumed in shifting the loading machine in the working place. Loss of some time is inevitable, but if the operator is experienced he will maneuver the machine to the most advantageous position while the cars are being shifted. The time of the four delays discussed, plus the time of actually loading coal and slate, accounts for 95.8 per cent of the time worked. This leaves 4.2 per cent of the total time distributed among four other minor delays. It is interesting to note that each shuttle car traveled approximately 6 miles during the shift. The average haul from the faces to the transfer station was 420 ft. Shuttle cars have proved to be particularly advantageous at this operation because a mine car of small capacity is used. The mine cars during the shift studied averaged 2.1 tons each, whereas the shuttle cars averaged 4.7 tons. Thus, if mine cars had been used to service the loading machine, twice as many car changes would have been necessary.

Summary

Although a number of problems in shuttle-car transportation must be solved, it has been accepted as an efficient system of service haulage in several mines of West Virginia. Probably the largest factor leading to its acceptance in these mines is the reduction of car-shifting time at the loading machine, with a resultant increase in production per machine. The latter statement, however, may be in controversy when a mine car as large or larger than the shuttle car is in use. Along with the elimination of track and trolley wire in the section, a greater flexibility of travel is obtained because the battery-powered cars are independent of track and wire. It is believed that greater safety is obtained by eliminating the noise of mine cars as well as the

coupling of mine cars. Other advantages claimed for this system include the reduction of peak power loads on gathering haulage, freedom of shuttle cars from direct effects of power-line failures, and additional flexibility gained by the ability of the shuttle car to follow the loading machine to any position, as shuttle-car movement is independent of track.

The greatest apparent disadvantage is the effect of roof and bottom conditions on efficiency. Shuttle-car mobility is more susceptible to roof conditions, although this is relieved somewhat by driving places on an angle when bad roof necessitates narrow places and close timbering. However, the car clearances still cannot be maintained as accurately as when mine cars and track are used.

Another serious drawback is the lack of cutting-machine capacity, although some companies are overcoming this with newer and faster machines. The cutting machines used, being mounted on caterpillar trucks, are so much slower than track-mounted machines that coal must be cut off shift.

The advantage of shuttle cars over mine cars is reduced with an increase in the capacity of the mine cars. When the capacity of the mine cars is greater than that of the shuttle cars, it may be questionable whether shuttle-car operation is profitable. When shuttle cars were first placed in the mines, they had some minor defects, most of which have been remedied. Some companies have completely rebuilt their cars, converting them to four-wheel drives instead of two and shortening the wheelbase. With a shorter wheelbase and four-wheel steering the cars are able to negotiate corners with greater ease and the hazard due to bad roof and to timber is reduced. Other disadvantages are less space to gob slate in haulways, reduction of battery power during the shift, and an increase in handling of coal.

The slow tramming of caterpillar equipment is almost sure to be remedied in the near future. A cutting machine with faster tramming speed and increased cutting capacity must be devised. As haulage is on rubber tires, why not mount all the equipment in that manner? A cutting machine on rubber tires may seem fantastic, but rubber-tired cars seemed as strange when their use was first suggested. The elevating conveyor, if equipped with rubber tires, would be much easier to move and could be transferred during the shift with almost no loss of time. A conveyor of this type is now available.

At the mines visited during this study, it has been proved that more coal can be produced by using shuttle cars than by the previous service haulage system. This shows that this system has its applications and the question before operators is: Can an increase in tonnage be obtained that will decrease the cost per ton?

Production-cost data for some of the mines of this study were not available and those that were available were not in forms to be comparable. Several items of cost are yet to be determined accurately, such as maintenance of equipment and depreciation.

As this system is still in the experimental stage, probably it will make even greater strides in the future than it has made in the past few years.

Hydraulic Brake for Mine Locomotives

By C. S. Allen*

(White Sulphur Springs Meeting, June 1941)

With increased coal production and mechanization of coal mines many transportation problems arise. The main objective is to bring the coal to the tipple or dump it as quickly as possible. Larger and faster haulage locomotives or two locomotives connected in tandem have been adopted to solve the problem.

Motor equipment, gears and control systems of locomotives to be placed on these hauls are carefully studied to meet these requirements, by an analysis of motor characteristic curves with the corresponding tractive effort curves, in order to determine the correct load for maximum efficiency and to determine the approximate schedule.

Control equipment is usually considered to be the controller and resistance. Sometimes accelerating curves are calculated in order to obtain smooth starting in the least possible time without slipping the wheels. Train control is very important, but why limit this to acceleration? Deceleration or retardation of the locomotive is a part of the train control and should be given as much consideration as the control for acceleration.

Types of Braking

Common braking systems for mine locomotives utilize screw-type mechanical brakes, dynamic braking and the air brake. For gathering and haulage locomotives, the screw-type mechanical brake is regarded as standard. A brake of this kind cannot be applied without much effort and lapse of time. Motormen, like other people, follow the path of least resistance, and therefore have their brakes ready for instant application—which means that the shoes are virtually dragging. Even then, a motorman bucks and plugs the motors to gain time and avoid as much physical effort as possible.

"Bucking of motors" consists in turning the reverse cylinder of the controller to parallel position corresponding to the opposite direction of travel while the locomotive is in motion, without applying power from an external source. When this occurs, both motors tend to act as series generators, because the armatures are revolving and there is residual magnetism in the field circuit. Owing to slight differences in motor manufacture, one of these will generate a slightly higher potential and will overcome the generated voltage of the second and tend to drive it as a motor in the reverse direction. This action retards the locomotive but is very severe on the segments of the reverse cylinder and on the motor commutators.

"Plugging motors" is the application of power to the motors corresponding to the opposite direction of travel. In order to "plug" the motors it is necessary first to "buck" them (which reverses the connections to the motors) and then to apply power. Theoretically, this should be done only in an emergency, but locomotives in mine service are frequently thus treated.

Dynamic braking is retardation of the locomotive by the motors acting as generators, which take no power from the line.

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* Jeffrey Manufacturing Co., Columbus, Ohio.

This is accomplished by making the proper connections to the armatures and fields, utilizing the residual magnetism to generate voltage. Both generators dissipate their energy through a resistor. Resistance is removed step by step as the control handle is moved toward the "full-on" position. The amount of braking effort depends on the speed of the locomotive and the amount of energy dissipated in the resistor. This type of braking is limited to track sections of long heavy grades. While a quick stop can be obtained by this method the braking effort is small as the generated voltage approaches zero.

Dynamic braking therefore is used successfully to limit the speed, but cannot be used effectively to make an emergency stop. This method increases the duty on the motors and resistance and this increases maintenance costs.

Air brakes have been installed on some locomotives when ample space was available, but this type of brake can be used only on large locomotives and tandem units.

As the weight of the locomotive increases, the difficulty of control also increases, and power brakes are essential. Many tests have been made to determine just where hand brakes should be displaced by power brakes, but each of these has so many variables that no decision has been reached. For instance, in a test on hand brakes, an operator who is strong and alert and is waiting for the signal loses no time in applying the brake when the signal is given, whereas with an operator who is very slow considerable time would be lost between the signal and the application of brakes. The pressure applied to the shoes would probably vary even more. About the only thing that could be determined would be the distance traveled when the brakes were applied with a given pressure. Track conditions also enter into the results, as well as the condition of the brake shoes and tires, and these conditions always vary. Actual experience has shown that 20-ton locomotives are on the dividing line, and power brakes are needed on all tandem units.

Hydraulic brakes can be installed on any locomotive, either gathering or haulage. For flexibility of installation, the various units are small and compact.

Stoppage of a Moving Train

A locomotive or a complete train will stop in a certain distance without the application of brakes, the retarding forces being air friction, friction of moving parts and friction of wheel flanges against the rail. In mine haulage, the air resistance can be neglected and the frictional resistance of moving parts and wheel against rail is only a small percentage of the frictional resistance offered by a brake shoe pressed against a revolving wheel. The amount of this resistance depends upon the pressure applied to the shoe and the coefficient of friction—between cast-iron brake shoes and steel tires this coefficient is about 25 per cent. The coefficient of static friction on clean dry rail between the wheel and the rail can also be taken as 25 per cent.

Maximum braking effort that can be applied to a locomotive is slightly under the coefficient of adhesion or static friction. When the friction of the shoes exceeds the friction of the wheel on the rail, the wheels will lock and slide, then the coefficient of friction will drop to about 8 per cent at 10 miles per hour and the locomotive will require a greater distance in which to stop than if the wheels were rotating.

All these considerations emphasize the importance of having the correct amount of brake-shoe pressure. Normal braking effort, therefore, is considered to be up to the point of wheel slippage on clean dry rail and emergency position to be the pressure required before the wheels would slip on sanded rails. Size of hydraulic brake cylinders is determined from the brake leverage and weight of locomotive.

The Hydraulic Brake

The hydraulic brake operates on the same principle as the air brake. The latter compares with the air tank on air equipment and will be discussed in detail later.

Motor, tank and pump unit as shown

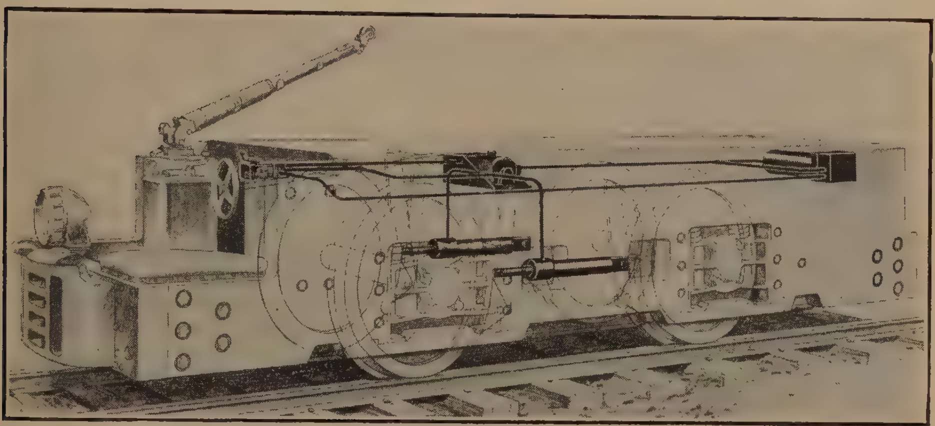


Fig. 1.—Haulage locomotive equipped with hydraulic brakes.

has a motor, compressor, air tank or reservoir, brake cylinder and control valve; the hydraulic brake uses a tank, motor and pump instead of motor, compressor and air tank. Brake cylinders and control mounted between the motors are analogous to motor and compressor of the air system. On air systems, the supply is drawn from the atmosphere, but on hydraulic, a closed system is used.

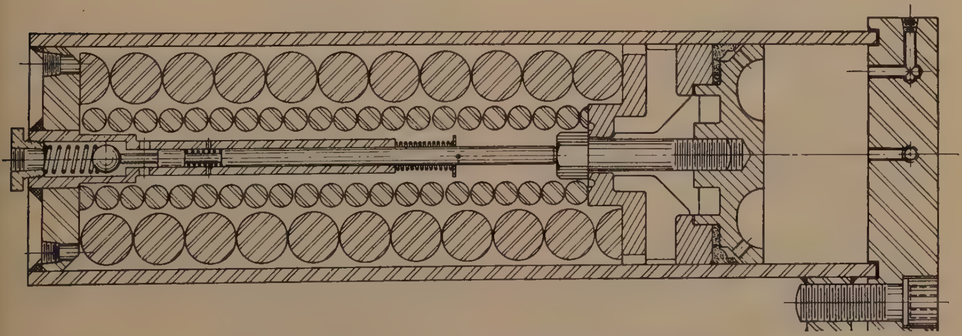
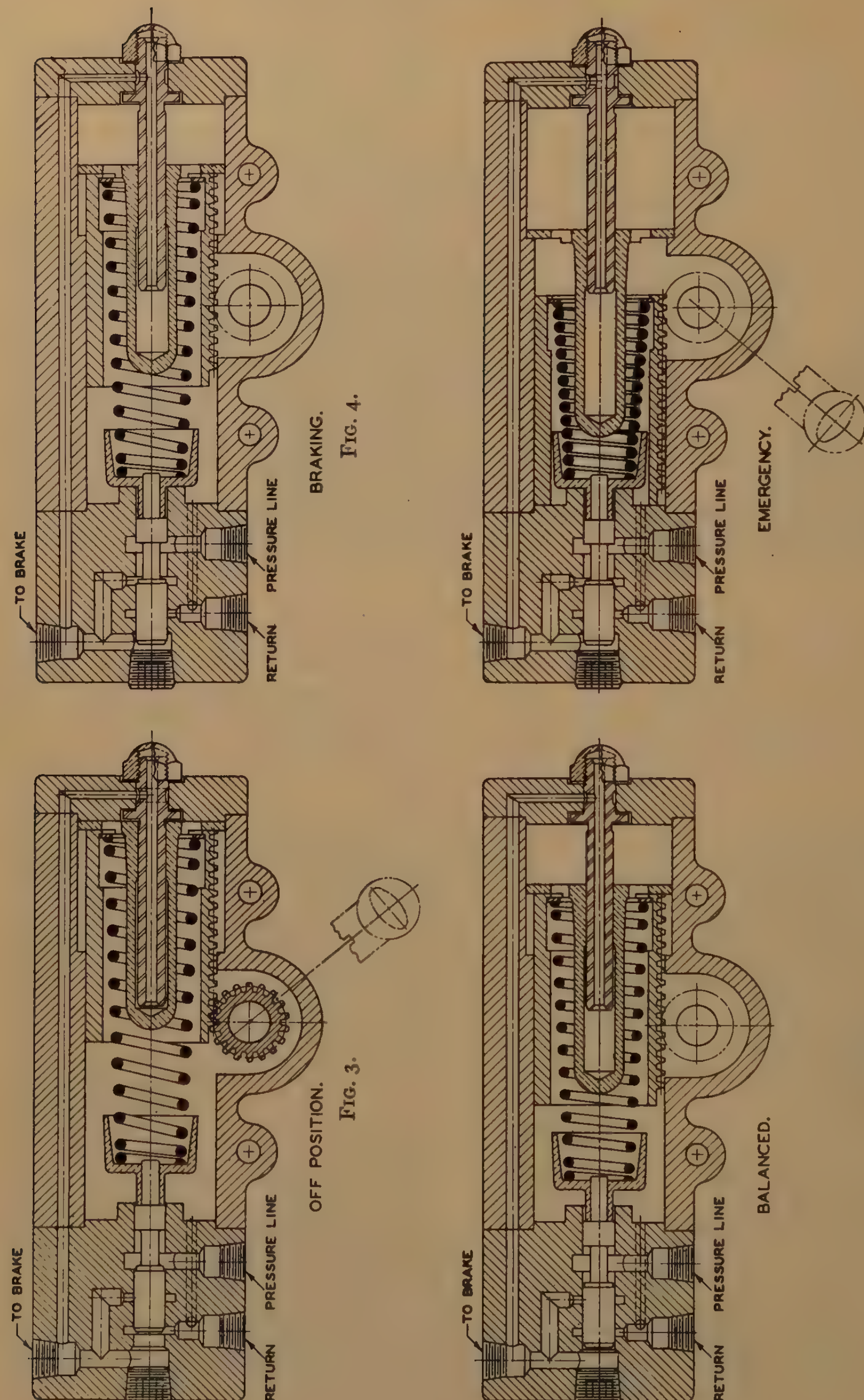


Fig. 2.—Accumulator with springs compressed.

valve are used on both, but of different design.

Fig. 1 shows a haulage locomotive equipped with hydraulic brakes. The hydraulic brake is about one-fourth the size of an air brake. The accumulator shown on the front end of locomotive acts as a reservoir for the oil under pressure. This

The accumulator (Fig. 2) is a steel cylinder with a 6-in. ground bore. The cup-packed piston is backed by two coil springs, which are shown fully compressed. The accumulator fully charged will hold 0.335 gal., which in case of power failure is sufficient for 10 to 20 brake applications. Over-all dimensions of this unit as shown



OFF POSITION.
FIG. 3.

BRAKING.
FIG. 4.

BALANCED.
FIG. 5.

EMERGENCY.
FIG. 6.

FIGS. 3 TO 6.—CONTROL VALVE IN DIFFERENT POSITIONS.

in Fig. 2 are 7½ in. square by 25⅛ in. long.

The oil enters the right side of the accumulator and as oil pressure builds up forces the spring-loaded piston back. When the piston moves back to the position shown, it pushes the stem (shown inside the small spring) and opens the spring-loaded ball by-pass valve. Oil is then by-passed to the tank at a circulating pressure of about 25 lb. per sq. in. The second line on the right side is connected to the control valve. A check valve is provided in the line from pump to accumulator, to prevent the piston from forcing oil out when the by-pass valve opens.

The motor, tank and pump unit are all mounted on a common base. The continuous motor is ¼ hp., 1750 r.p.m., 250 or 500 volt. The load is intermittent, as the pump does not operate against high pressure when the by-pass valve is open. The pump is a small axial-piston type having high efficiency, delivering ⅓ gal. per min., and does not require packing because it is completely submerged in oil inside the tank. Over-all dimensions of this unit are 9¼ in. wide, 11⅛ in. high and 20½ in. long.

Fig. 3 shows the control valve in the off position. When the control handle is moved to braking position as shown on Fig. 4, the rack, actuated by the geared handle, moves to the left, compressing the spring, which forces the valve to the position shown. Oil then enters through the pressure line to the brake cylinders. Connected in parallel with the line to the cylinders is a small opening drilled in the valve body to the back side of the rack. When pressure builds up in the cylinders it is transmitted to the valve rack and at the same time to the back side of the valve. If the handle is left in this position the pressure on the valve closes the port (Fig. 5). The valve will remain in this position because the pressure on the valve and rack counterbalances the spring pressure. Any desired pressure on the cylinders can be obtained by turning the handle, which moves the rack either right or left. If the braking is too severe, the handle is moved toward the "off" position. When this occurs, the pressure on the spring is decreased, which unbalances the valve; as the pressure is greater on the valve proper, it moves to the right, permitting oil to return to the tank. When the pressure on the valve becomes equal to the spring pressure the valve closes the return port.

All through the normal braking position —that is, from zero to 500 lb. per sq. in. on the cylinders—any movement on the handle changes the spring pressure on the valve. If it is necessary to go to the emergency position (Fig. 6), the spring is fully compressed and must be held in this position. When the handle is released, pressure on the valve forces the spring seat to the right because the spring is designed only for working pressure of 500 lb. per sq. in. Cylinder pressure for a given control-handle setting can be changed by varying the size of spring. Over-all dimensions of this unit are 9¾ in. long, 2⅞ in. wide and 5⅜ in. high.

Discussion on the valve mechanism and operation applies to a balanced hand-operated valve. By using a gear and rack assembly it can very easily be converted to a foot-operated valve. To do this, it is necessary to have the foot-pedal spring loaded so that when the foot pressure is removed the valve will return immediately to the off position. This type of control was designed for use on a gathering locomotive, so that the operator could have both hands free to operate the controller.

It is well to bear in mind that the diameter or bore of brake cylinders is determined from the weight and brake linkage of the locomotive. Actual applications have demonstrated that it is necessary to calculate the force required to slide the wheels on clean dry rail and then to increase this 35 per cent, to take care of the various

losses in brake linkage and to correct the fact that nominal weights are given instead of actual.

when the cylinder operates the levers pivot on the equalizing lever pin. Fig. 8 illustrates another type of mounting, having a slot

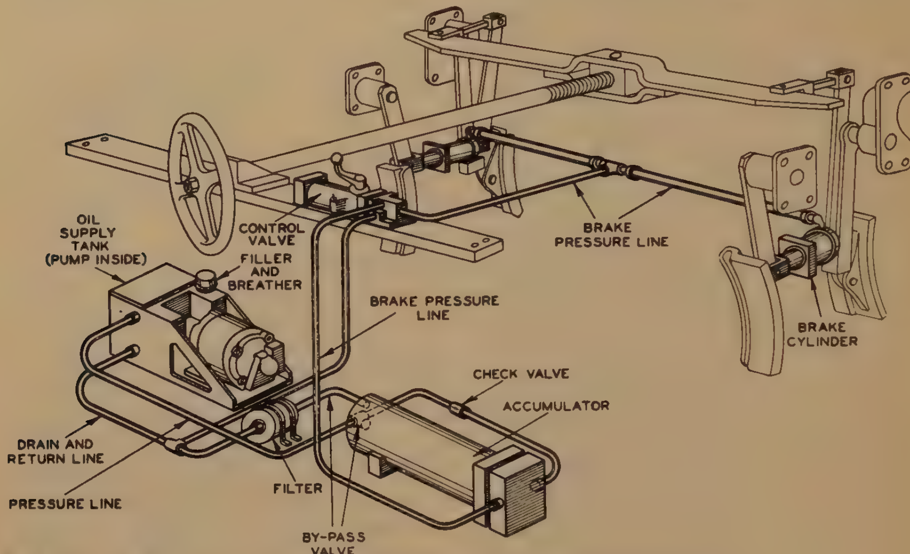


FIG. 7.—ONE TYPE OF MOUNTING OF HYDRAULIC BRAKES.

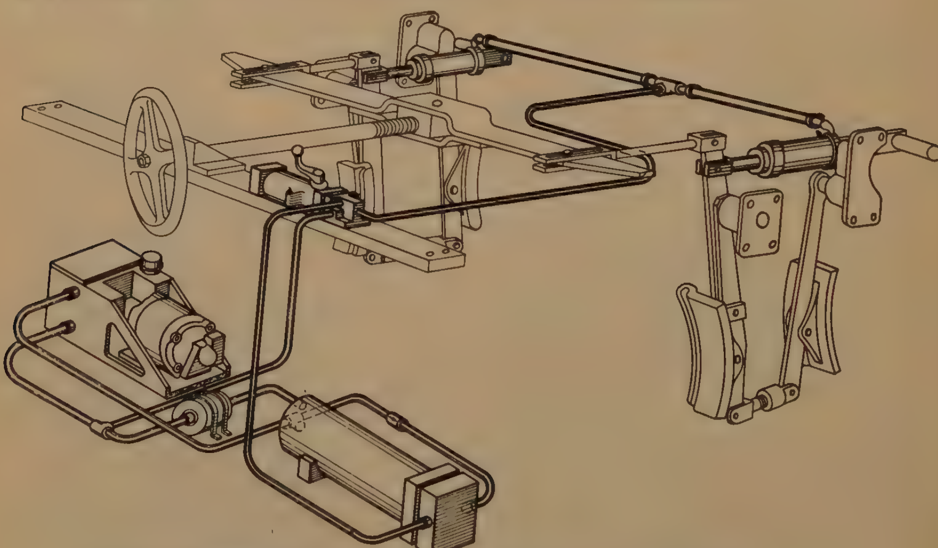


FIG. 8.—MOUNTING OF HYDRAULIC BRAKES HAVING SLOT ON EQUALIZING LEVER.

on the equalizing lever so that the cylinder can operate independently of the screw brake, which is retained for a parking brake.

Installation of hydraulic brakes on single or tandem units is a simple matter. Fig. 7 illustrates one type of mounting. The brake cylinder is placed between the shoes and

Fig. 9 shows the simplified piping diagram for hydraulic brakes.

Installation on Tandem Locomotives

Permanent tandems use one tank, pump and motor unit, one accumulator and valve and two sets of cylinders with necessary tubing and hose. It is not necessary to shift any of the electrical equipment to provide room for these units and as the total weight, except cylinders, is only 295 lb. no weight need be added for counterbalance.

Separable tandems require an accumulator, pump and tank unit with a valve on each unit along with the cylinders. When they are used in tandem it is necessary only to disconnect the hose lines from the valve on the secondary and reconnect to the hose lines from the primary.

Several installations are in actual service, the largest number being in West Virginia, where grade conditions make their incorporation necessary. Nearly all these units are tandems and are accepted with as much approval as the hydraulic brake on an automobile.

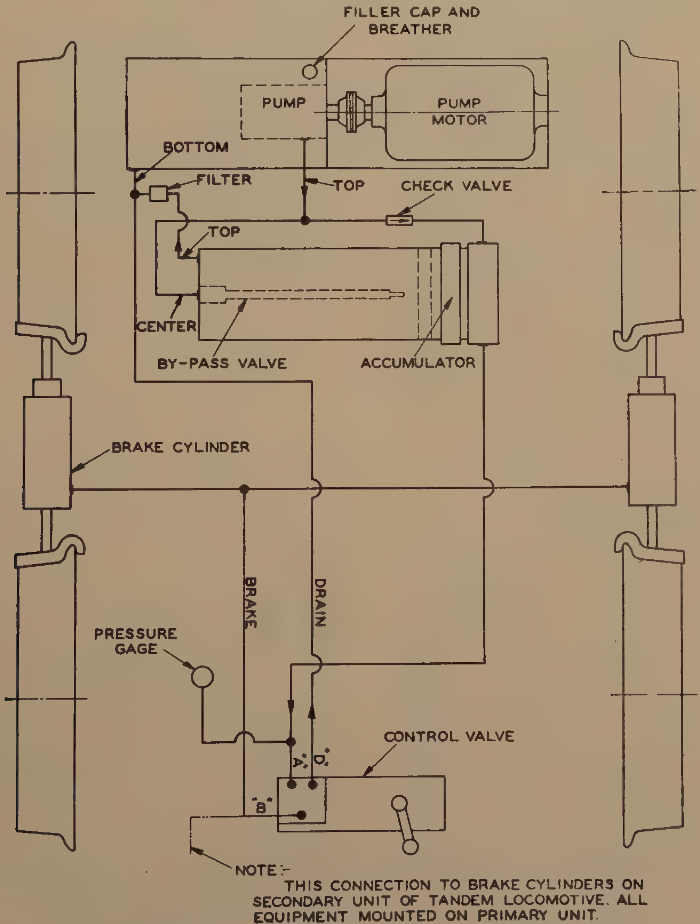


Fig. 9.—Simplified piping diagram for hydraulic brake.

Savings in Maintenance

Size, simplicity and demand for a suitable brake on locomotives have been stressed,

but not the savings in maintenance. These are:

1. It is no longer necessary to "buck" the motors, which decreases the maintenance cost on armatures, brushes, brush rigging, reverse cylinder and fingers except when the motors originally were arranged for antibuck.

2. Motors no longer need "plugging," and this decreases maintenance costs on controller fingers and segments, resistance, resistance leads and possible flash-overs caused by excessive voltage at the brushes.

3. Decreased tire wear.

4. Much time is saved in applying and releasing the brakes, which enables the motorman to maintain a higher average speed.

DISCUSSION

(*J. E. Tobey presiding*)

C. Scholz,* Charleston, W. Va.—The application of hydraulic brakes to locomotives is a decided forward step and should be used more extensively. I believe this type of braking could be used to advantage on mine cars. In mine locomotives the adjustment of mechanical brakes is a source of constant expense and, unless very well maintained, the cause of expensive accidents.

Bucking of motors is expensive, and should be avoided.

* Consulting Engineer.

Physical Properties of Coal and Associated Rock as Related to Causes of Bumps in Coal Mines

By Charles T. Holland,* Junior Member A.I.M.E.

(New York Meeting, February 1941)

In connection with the problems of bumps in coal mines, much has been written concerning the manner in which roof action and methods of mining enter into the pressure effects observed but little has been written upon the relationship between the physical properties of coal and associated rocks and the cause of bumps. The question of the strength of bituminous coal, of its elasticity, and the energy it can absorb and store has been neglected to a considerable extent in connection with this problem.

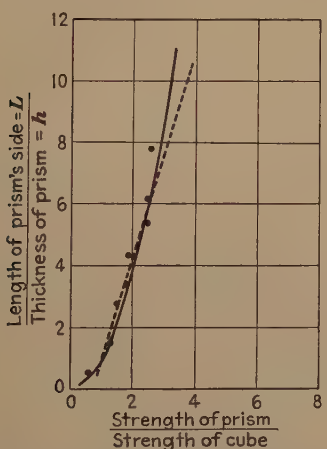


Fig. 1.—Relation between dimensions of test specimen and strength developed.

Solid line represents conclusions of Scranton Engineers' Club. Dashed line represents Bunting's conclusion.

Strength of Coal

It is generally agreed that one of the necessary conditions for bumps is a strong coal.[1] Therefore, since the strength of coal is an important factor, a study of this property should be made.

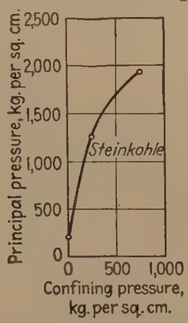


Fig. 2.—Increase in strength of coal due to application of confining pressure. (After Müller.)

Several investigators have conducted laboratory experiments upon the strength of coal, using small cubical or rectangular test pieces. An attempt will not be made to go into all of these investigations, but they are briefly summarized in Table 1. Experiments of the U. S. Bureau of Mines[8] and of Lawall and Holland (pp. 27 and 34 of ref. 9), however, indicate that regardless of whether the coal is a soft and weak semibituminous (Pocahontas No. 3) or a hard and strong bituminous (Stockton) coal, the maximum ultimate unit stress developed in laboratory tests is higher in small cubes than in larger cubes of the same coal.

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* School of Mines, West Virginia University, Morgantown, West Virginia.

[1] References are at the end of the paper.

Further tests by the U. S. Bureau of Mines indicate this property to be true for small coal pillars in a mine. Table 2 is reproduced to show the results of these tests. It shows that of the pillars in which the ratio of lateral dimension to height is unity the pillar having the largest lateral area, 4078 sq. in., developed the lowest ultimate unit stress under load.

TABLE 1.—*Strength of Coal as Indicated by Laboratory Tests on Small Specimens*

Observer	Reference	Location	Size of Test Specimen	Kind of Coal	Strength Observed
Müler	2	Germany		Steinkohle	Heinitz bed, 210 kg per sq. cm. Schuchman vein, 165 kg per sq. cm.
Louis	3	England	3 to 4-in. cubes		0.25 to 2.84 ton per sq. in., load applied perpendicular to bedding 0.31 to 1.70 ton per sq. in., load applied parallel to bedding
Penman	4	India	Varied		1000 to 4600 lb. per sq. in., direction relative to bedding not stated
Daniels and Moore	5	Pennsylvania	Varied 2-in. cubes to 4 × 4 × 12-in. column	Anthracite	831 to 3870 lb. per sq. in.
Daniels and Moore	5	Pennsylvania	Varied	Bituminous	700 to 1538 lb. per sq. in.
Scranton Engineers' Club.	6	Pennsylvania	2 × 2 × 1 in. 2 × 2 × 2 in. 2 × 2 × 4 in.	Anthracite	1744 to 7009 lb. per sq. in. for 2-in. cubes
Talbott	7	Illinois	11 to 15-in. cubes	Bituminous	1000 lb. per sq. in. to 2170 lb. per sq. in. Average, 1490 lb. per sq. in.
U. S. Bureau of Mines	8	Pennsylvania (Pgh. bed)	2½ to 4-in. cubes	Bituminous	2486 lb. per sq. in., load applied perpendicular to bedding
Lawall and Holland	9	W. Virginia	3-in. cubes	Bituminous and semibituminous, 22 beds	1163 to 6841 lb. per sq. in., load applied perpendicular to bedding 823 to 5009 lb. per sq. in., load applied parallel to bedding

TABLE 2.—*Crushing Strength of Pillars*[10]

Pillar No.	Area in Compression, Sq. In.	Ratio of Lateral Dimension to Height	Maximum Total Load, Lb.	Strength, Lb. per Sq. In.
2	1,009	0.50	505,600	500
4	2,057	0.75	1,236,900	600
7	4,078	1.00	2,835,000	695
6	1,040	1.01	920,700	885
5	1,010	1.03	930,600	920

As experimental investigations reveal that the strength of coal in compression is so markedly influenced by the absolute size of the test specimen, experimental values of strength cannot be applied in estimating the strength of coal pillars. Also, the fact that coal-mine pillars have large lateral dimensions compared to their thickness enters into the problem of the strength of coal in those pillars. Tests made for the Scranton Engineers' Club on 423 samples of anthracite having dimensions of 2 by 2 by 1 in., 2 by 2 by 2 in. and 2 by 2 by 4 in. indicated that: "In general, other things being equal, the crushing strength of mine pillars would vary inversely as the square root of the thickness of the bed" (p. 78 of ref. 6). Bunting,[11] after analyzing the results of tests on 647

specimens having different ratios of lateral dimension to height, advanced the rule of

$$\frac{\text{Strength of Prism}}{\text{Strength of Cube}} = 0.70 + 0.30\frac{b}{h}$$

in which b = least lateral dimension and h = height.

Lawall and Holland investigated the effect of varying the thickness while the lateral dimensions were held nearly constant at 3 in. on each side. The results of those experiments were extremely erratic when individual results are considered. abruptly; however, some of the specimens having an $\frac{l}{h}$ ratio of 4.9 and higher did not fail abruptly. This action, it seems, may have a special significance. Table 3 lists the particulars concerning the test specimens that did not fail abruptly.

These results are interesting because the test specimens have the same $\frac{l}{h}$ ratios that often are found in coal-mine pillars. While probably it would be incorrect to say that the specimens did not fail, yet

TABLE 3.—*Description of Specimens That Did Not Fail Abruptly*

Name of Bed	$\frac{l}{h}$	$\frac{\text{Strength of Prism}}{\text{Strength of Cube}}$ $\frac{p}{c}$	Average of 3-in. Cubes, Lb. per Sq. In.	Maximum Load, Lb.	Maximum Stress, Lb. per Sq. In.
Beckley[a]	4.9	14.7+	1,163	200,000	26,500
Island Creek[b]	6.8	4.56+	4,725	194,250	21,600
Island Creek[c]	12.0	8.30+	4,725	353,000	39,200
Coalburg[d]	11.5	6.85+	6,861	400,000	46,300
Hernshaw[e]	9.61	4.72+	4,727	201,000	22,300

[a] This specimen never completely failed in the sense that it stopped taking weight; at a total load of 60,000 (on 7.56 sq. in.) it apparently took load more slowly; compact when taken out of testing machine, but very fragile. Test discontinued because bearing plate failed.

[b] This specimen did not completely fail but bearing plate failed. Area 9 sq. in.

[c] Bearing plate bent but coal was still intact. Area 9 sq. in.

[d] Coal did not fail in the sense that it stopped taking load or in that there was a noticeably abrupt change in the rate of taking load. Tool-steel bearing plate bent. Dimension of specimen before testing, 1/4 by 2 7/8 by 3 in.; after testing, 3 3/8 by 3 1/4 by approximately 1/4 inch.

[e] Coal did not fail in the sense that it stopped taking load or abruptly changed its rate of taking load, but the coal acted to shove 3/8 in. thick bearing plate into 1 in. hole in head of testing machine. The imprint of this hole was also clearly visible on the coal specimen.

However, if results of the individual tests having an $\frac{l}{h}$ (lateral dimension l divided by thickness h) of between 1 and 2 are averaged and those having an $\frac{l}{h}$ of between 2 and 3, and so on up to an $\frac{l}{h}$ of about 8, and the corresponding ratios of strength of prism to strength of cube are averaged, a more uniform set of results is obtained. These average results are shown in Fig. 1 by the small circles. The curves drawn represent the conclusions of Bunting and of the Scranton Engineers' Club. Both of these curves fit the results very well up to an $\frac{l}{h}$ of about 8 for the specimens that failed there was no point during the application of the load when it could be stated definitely that failure occurred. The bending of tool-steel bearing plates, denting of soft-steel bearing plates, and the distinct imprint of the round hole on the coal specimen suggest that the coal was forced into a plastic state. Whenever these specimens were examined after loading they were intact (except around the edges) but were rather fragile. When the specimen was broken, shear surfaces apparently could be seen, but they were very smooth and polished (slickensided).

It is important, perhaps, to note that the Beckley coal, having a crushing strength as a 3-in. cube of 1163 lb. per sq. in. was

forced into gradual failure at an $\frac{l}{h}$ ratio of less than 4.9 but greater than 2.87. The Island Creek coal, having a crushing strength of 4725 lb. per sq. in., failed abruptly when the $\frac{l}{h}$ ratio was as high as 6.7, but was forced into gradual failure at an $\frac{l}{h}$ ratio of 6.8. The Hernshaw coal, having a crushing strength of 4727 lb. per sq. in. failed abruptly at an $\frac{l}{h}$ of 6.85 but did not fail abruptly when the $\frac{l}{h}$ ratio was raised to 9.61. The Coalburg coal, having a crushing strength of 6861 lb. per sq. in., failed abruptly when the $\frac{l}{h}$ ratio was 7.82 but when the value of this ratio was increased to 12 the specimen was forced into gradual failure. These results, although not conclusive, suggest that the stronger a coal, the more difficult it is to force it into failure by flow or by squeezing. The fact that the Island Creek and Hernshaw coal had about the same strength in 3-in. cubes, but that the Hernshaw did not fail by squeezing until the $\frac{l}{h}$ ratio was somewhere between 6.85 and 9.61 while the Island Creek failed by squeezing when the ratio was between 6.7 and 6.8, suggests that the demarcation between abrupt failure and squeezing is not sharply defined but gradually passes from one type of failure to the other. If these results can be applied to coal-mine pillars, the possibility is indicated that the stronger the coal is in a mine pillar, the wider a pillar must be relative to its height to prevent abrupt failure. They also suggest that in two pillars having the same $\frac{l}{h}$ ratio, one might fail abruptly and the other one squeeze; this being true, of course, only when the $\frac{l}{h}$ ratio is close to the value at which abrupt failure passes into gradual failure.

The factor that causes the increase in strength as the $\frac{l}{h}$ ratio increases must be the effect of the confining pressure due to friction between the faces of the coal specimen and the bearing blocks of the testing machine. It is interesting to note that Müller (p. 1611, ref. 2) tested coal having a uniform confining pressure applied. His results are reproduced in Fig. 2. The figure indicates that a comparatively small side pressure of 750 kg. per sq. cm. (10,650 lb. per sq. in.) increased the crushing strength from 200 kg. per sq. cm. (2840 lb. per sq. in.) to 1930 kg. per sq. cm. (27,400 lb. per sq. in.). Müller's results demonstrate that with comparatively small side pressure the crushing strength increases very much and that this effect is not so rapidly increased after the confining pressure is increased above a certain value. He notes that the specimen was not forced into a plastic state by the side pressure used. It should be noted in this respect that Griggs[12] did not find material increases in the plasticity of marble and limestone until the confining pressures exceeded 4000 atmospheres. While the two experimenters used different methods, and their results do not agree entirely as to the effect produced upon the ultimate strength of rocks by the confining pressure, they do agree that it takes a comparatively high confining stress to force brittle materials into failure by plastic deformation rather than by brittle failure.

The maximum stress a pillar will stand may be investigated by determining the bearing strength of small areas. This method has been used by the Bureau of Mines[13] on pillars in the Pittsburgh bed at Bruceton, Pa. In this investigation it was found that as the area under load was increased, the unit load it would support without failure decreased. It was determined, however, that with 200 sq. in.

under test the area would fail when the stress was increased to 5100 lb. per sq. in. Extrapolation of the results seems to indicate that a very large area of this coal would have an ultimate bearing strength somewhat in excess of 4000 lb. per sq. in. These investigations were made by applying the load parallel to the bedding planes of the coal rather than perpendicular to these planes. This, however, should not produce a great amount of difference in the results obtained. A consideration of the results described by the authors indicates that the ultimate bearing strength of the coal tested was reached when the coal was forced into flow. This view is based upon the reversal of the deformation around the area undergoing tests after the load was increased beyond a certain point.

The experiments upon thin specimens and the bearing strength of coal do not give information directly upon the strength coal may develop in a mine pillar. The reasons for this are: (1) presence of joints and slips in mine pillars; (2) the bearing plates used in these experiments were of steel, and consequently very stiff as compared to a mine roof and floor, so the stress distributions are not the same; (3) it is not certain that linear similitude in the thin specimens is the same as mechanical similitude, because it is not possible to deduce the mechanical laws applying to the relationship between the dimensions and strength of a pillar; (4) in the bearing-strength experiments, the coal directly under compression is confined on all sides—this condition probably is realized in the center of large pillars but in a mine each pillar has four sides free. The following conclusions, however, may be made:

1. The ultimate bearing stress of coal or the stress under which it deforms by plastic flow or by squeezing is the maximum stress a pillar can develop.

2. This ultimate bearing stress is probably rather closely related to the crushing strength of coal as determined by tests on small specimens; that is, the bearing strength of a coal having a low strength when tested as a 3-in. cube will be less than the bearing strength of a coal having a high strength when tested as a 3-in. cube.

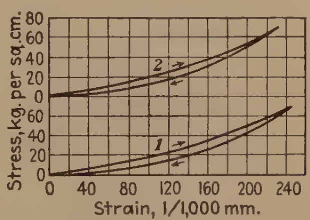

FIG. 3.—STRESS-STRAIN CURVES FOR COAL FROM THE SCHUCKMAN BED. (AFTER MÜLLER.)

3. If the coal composing the pillar is strong the $\frac{l}{h}$ ratio will have to be larger to prevent abrupt elastic failure under a uniform load than if the coal composing the pillar is weak.

4. There will be a range in the $\frac{l}{h}$ ratio over which a pillar may fail either abruptly or by squeezing.

Elastic Properties of Coal

Daniels and Moore (p. 263 of ref. 5) have made a few tests on the stress-strain characteristics of bituminous coal from the Pittsburgh bed and on anthracite, but they do not give any values other than to reproduce the curves obtained from tests on coal when using an autographic testing machine. Müller made several complete tests upon two German coals and associated rocks. Figs. 3 and 4 show his results. Fig. 3 shows the load reactions of coal from the Schuckman bed and Fig. 4 the influence of load upon the modulus of elasticity of coal and associated rocks. Müller concluded that coal shows an elastic modulus of 0.2 to 0.6×10^{11} in c.g.s. units. He also concludes that coal shows a large permanent set and a very much greater change in length for the same load than other rocks do in contact with the coal seam.

Lawall and Holland[9] conducted tests upon the stress-strain and elastic properties of samples from 21 coal beds in West Virginia. Their findings may be

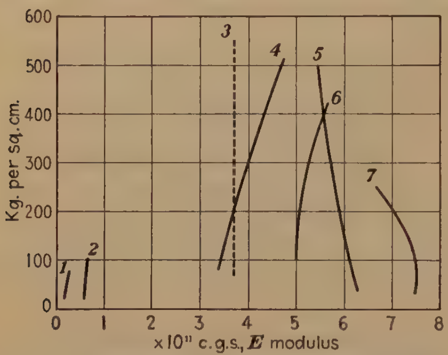

FIG. 4.—EFFECTS OF STRESS UPON ELASTIC PROPERTIES OF ROCKS. (AFTER MÜLLER.)

1. Steinkohle from Schuckman bed.
2. Steinkohle from Heinitz bed.
3. Slate; load applied perpendicular to cleavage.
4. Sandstone; load applied perpendicular to cleavage.
5. Slate from footwall.
6. Sandstone; load applied parallel to cleavage.
7. Slate from hanging wall.

summed up as follows: (1) All the coals tested showed considerable elasticity under short-time stresses, even when the stresses were carried close to the point of failure. (2) Nearly all the stress-strain graphs were curved lines over the first part of the curves, and many over the entire curve. (3) When the secant was drawn between the origin and a point equal to one-half the ultimate strength of the coal, the secant modulus of elasticity varied between 235,000 and 615,000 lb. per sq. in. These values of the elastic properties were obtained when the load was applied perpendicularly to the bedding planes of the coal. When the load was applied parallel to the bedding, the secant modulus varied between 105,000 and 800,000 lb. per sq. in. Fig. 5 illustrates the stress-strain and elastic properties of coal.

Greenwald and associates,[10] testing coal in small pillars in the experimental mine, obtained stress-strain curves (Fig. 6). None of these graphs are straight lines and the authors state that the modulus of elasticity as determined by a secant varied between 105,000 and 365,000 lb. per sq. in., depending upon the stress values between which

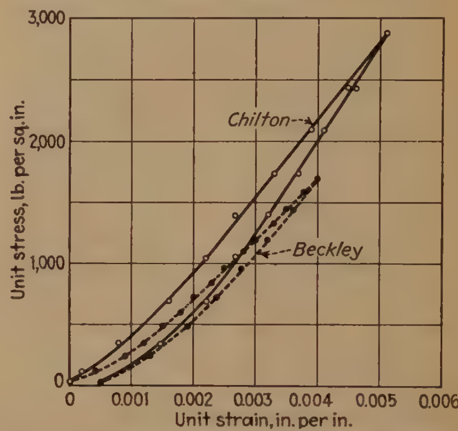

FIG. 5.—STRESS-STRAIN CURVES OF CHILTON AND BECKLEY COAL. (LAWALL AND HOLLAND.)

the secant was drawn and the pillar tested. However, they did not release the load to determine the permanent set or the hysteresis loop. They say that a large permanent set and wide hysteresis loop would be expected. An important fact to notice in this case is that the load was applied over a period of several days rather than over a few minutes, as in laboratory tests.

VARIATION OF ELASTIC MODULUS WITH LOAD

Since the stress-strain curves for coal are not straight lines, the modulus of elasticity varies with the load. To illustrate the variation in the elastic properties of coal, the curves in Fig. 7 were constructed from Lawall and Holland's test results. The curves shown in the figure indicate that the elastic modulus increased steadily until a stress of approximately 2000 lb. per sq. in. was attained. From this point up to a stress as high as the coal was tested, the elastic modulus remained nearly constant.

In the tests on the small pillars by Greenwald, the modulus of elasticity steadily increased until the stress reached approximately 500 lb. per sq. in., then from this

point on up to the ultimate stress, the modulus of elasticity decreased (Fig. 8). Müller's investigation revealed that as high as he carried his stresses, the modulus of elasticity of coal continued to increase (Fig. 4).

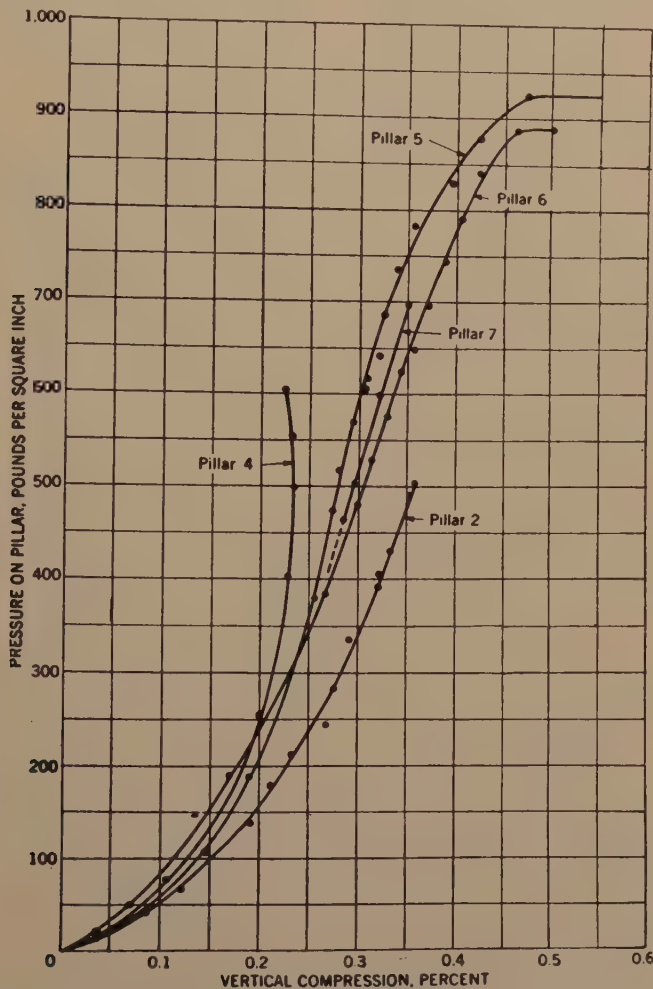

FIG. 6.—VERTICAL COMPRESSION OF PILLARS 2, 4, 5, 6 AND 7.
From U.S. Bur. Mines *Tech. Pub.* 605.

VARIATION IN STRESS-STRAIN PROPERTIES BETWEEN DIFFERENT POINTS IN SAME SEAM

Also, coal probably shows considerable divergence in elastic properties and stress-strain properties at different locations in the same seam. These may be grouped as regional differences and local differences. The regional differences may be due to differences in the rank of the coal and possibly differences in the coal-forming materials themselves.

In addition to these regional variations, differences in the stress-strain characteristics may be expected in the same mine and in the same pillars. Fig. 9 shows the stress-strain characteristics of three samples of the Pittsburgh coal taken in the same mine, only a few hundred feet apart.[14] These

curves indicate that at a stress of 2000 lb. per sq. in. the specimens of sample No. 1 deformed 71.5 per cent as much as the specimens from sample No. 3 and 75 per

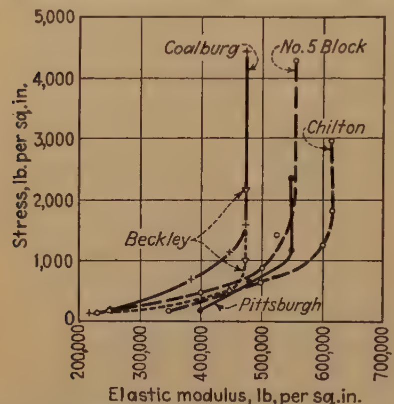

FIG. 7.—VARIATION OF ELASTIC MODULUS WITH INCREASING STRESS IN COAL.

cent as much as the specimens of sample No. 2. It should be noted in Fig. 6 that pillar 7 shows about 82 per cent as much compression as pillar 2 when both are

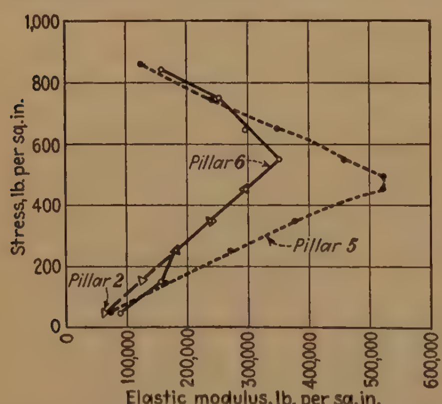

FIG. 8.—VARIATION OF ELASTIC MODULUS IN SMALL MINE PILLARS TESTED BY GREENWALD.

These curves give elastic modulus of pillars calculated from slope of stress-strain curves at a particular stress.

under a load of 500 lb. per sq. in. These two pillars were only 45 ft. apart. The two pillars were of different dimensions but experiments upon 3-in. cubes and 6½-in. cubes of the same kind of coal seem to indicate that the dimensions of the test pieces may not influence the stress-strain properties appreciably.

In addition to the variation between

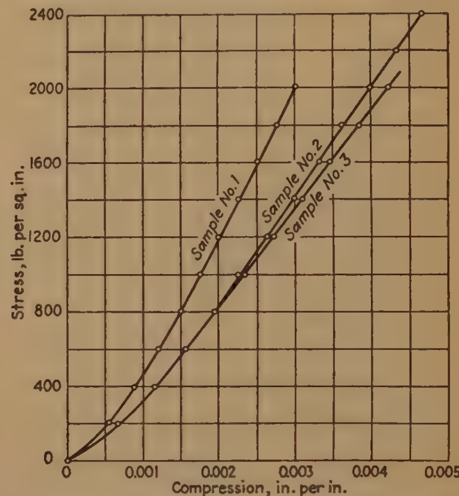

FIG. 9.—AVERAGE STRESS-STRAIN CHARACTERISTICS OF COAL SAMPLES TAKEN AT DIFFERENT POINTS IN SAME MINE.

different lateral points in the same seam, probably there is a difference in the stress-strain characteristics in the different parts of the vertical section of the bed. As an

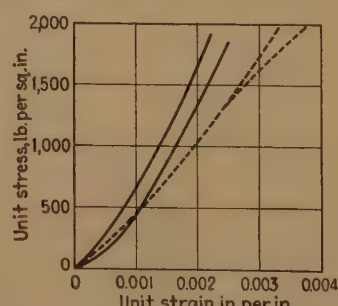

FIG. 10.—VARIATION OF STRESS-STRAIN CHARACTERISTICS OF COAL SPECIMENS TAKEN FROM DIFFERENT POINTS IN VERTICAL SECTION AT PARTICULAR LOCATION IN PITTSBURGH BED.

indication of the difference to be expected, the curves of Fig. 10 were drawn. These curves show the stress-strain characteristics of specimens cut from samples taken from various sections of the Pittsburgh bed at

the same location. Greenwald (p. 16 of ref. 10), in his experiments upon small mine pillars, found the top half of the seam compressed about 10 per cent more than the bottom at a pressure of 700 lb. per sq. in., but he says "before significance can be attached to this difference, confirming evidence will be necessary." These results are shown in the curves of Fig. 6, pillars 5 and 6.

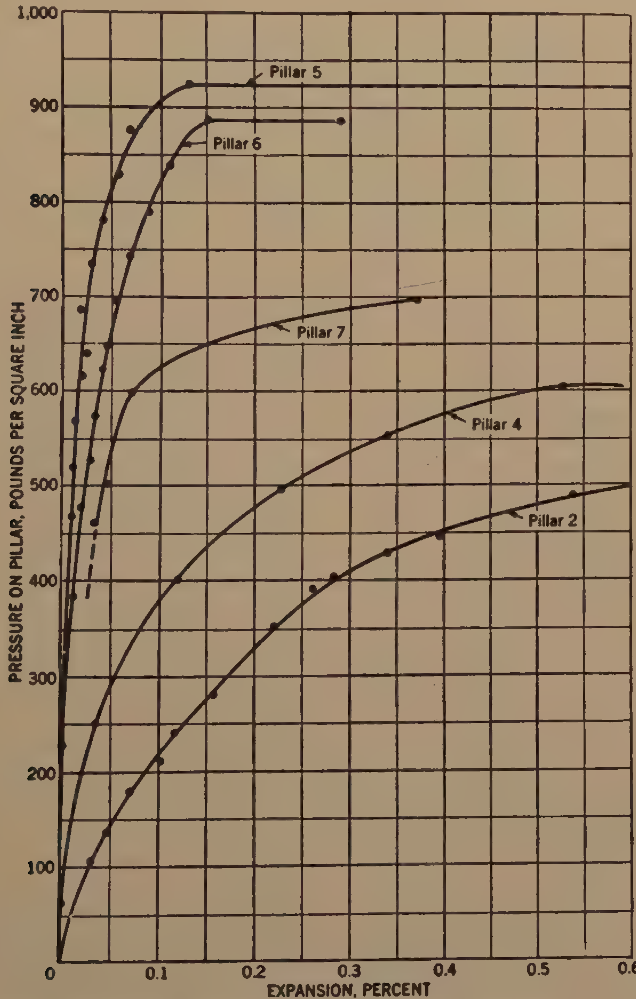

FIG. 11.—AVERAGE EXPANSION OF PILLARS AT MIDHEIGHT AS INDICATED BY OUTWARD MOVEMENT OF FACES.
From U.S. Bur. Mines *Tech. Pub.* 605.

If the various sections of coal beds have different stress-strain characteristics under compression, it is logical to expect these members to show differential lateral extensions as the compressive load on the pillars is increased. It seems, however, that there is no definite information on this subject. Greenwald's results indicate that the lateral expansion of pillars due to a compressive load does vary at different localities in

the same bed. The curves of Fig. 11, for example, show that at a stress of 885 lb. per sq. in. pillar 5 expanded approximately half as much as pillar 6. Also, the fact that pillars 5 and 6 were cut out of the bottom and top halves of the coal bed may have accounted for some of this difference.

TABLE 4.—*Energy Stored and Absorbed by One Specimen Tested from Each of Several Coal Beds in West Virginia*

No. of Specimen	Name of Coal	Crushing Strength of Specimen, Lb. per Sq. In.	Strain Energy, In-lb. per Cu. In. to Produce a Stress Equal to One-half Ultimate Strength	Ratio Maximum Strength Tested to Ultimate Strength	Strain Energy Stored and Released, In-lb. per Cu. In.[e]			Strain Energy Absorbed	Strain Energy Released / Energy Stored
		[a]			[b]	[d]	[e]		
1A	Pittsburgh	2,532	2.16	0.74	1,860	2.96	2.51	0.45	0.85
2	Sewickley	4,980	3.8	0.50	2,500	4.1			
3	Pocahontas No. 3	1,865	0.96	0.70	1,305	2.09	1.75	0.34	0.84
4	No. 5 Block	5,920	8.1	0.73	4,350	17.15	14.43	2.72	0.84
4B	No. 5 Block	1,892	1.54	0.51	967	1.62	1.28	0.34	0.79
5	Hernshaw	4,800	4.92	0.79	3,810	11.46	10.93	0.53	0.95
6	Douglas (Red Ash)	1,745	1.13	0.70	1,221	1.87	1.65	0.22	0.88
7	Welch	1,575	1.01	0.59	928	1.28	1.03	0.25	0.80
8	Pocahontas No. 5	1,810	0.99	0.74	1,332	1.97	1.78	0.19	0.90
9	Beckley	1,898	1.08	0.90	1,695	3.04	2.58	0.46	0.85
13	No. 2 Gas	3,180	2.71	0.84	2,670	6.80	5.74	1.06	0.84
14	Alma	3,310	2.70	0.58	1,925	4.31	3.30	1.01	0.77
16	Powellton	1,730	1.38	0.77	1,331	3.29	1.46	1.83	0.44
20	Waynesburg (Upper Bench)	4,030	3.69	0.66	2,650	6.23	5.29	0.94	0.85
21	Bakerstown	2,670	1.5	0.52	1,395	1.6		No data	
22A	Upper Freeport	2,380		0.47	1,110	1.42	1.22	0.20	0.86
25	Island Creek	5,140	5.72	0.66	3,400	9.30	7.92	1.38	0.85
27	Chilton	4,600	4.60	0.62	2,870	6.80	6.05	0.75	0.89
28	Sewell (Davy)	2,300	1.81	0.57	1,305	2.05	1.58	0.47	0.77
29	Coalburg	6,430	11.91	0.64	4,100	18.25	14.61	3.64	0.80
30	Winifrede	5,950	8.72	0.69	4,070	13.66	11.74	1.92	0.86
31	Stockton	7,260	12.2	0.57	4,170	15.7	12.4	3.3	0.79

[a] All specimens approximately 3-in. cubes.
[b] Stress in pounds per square inch. This value of stress was the highest value at which a strain measurement was made.
[c] These figures are representative but some specimens showed slightly more energy released than was applied. The cause of this is attributed to one or more of the following reasons: (1) Nonuniform stresses in the specimen (supported by some evidence); (2) inaccuracies in measuring strains; (3) energy released from coal that was stored by geologic forces.
[d] Inch-pounds of energy per cubic inch of specimen required to produce this stress.
[e] Strain energy released per cubic inch of specimen when stress was reduced to zero.

STRAIN ENERGY STORED AND ABSORBED BY COAL

From the experimental results of Lawall and Holland, a study of the energy absorbed, released and stored by 23 coal samples from 21 different beds has been made. These results are shown in Fig. 12. Curve *A* shows the energy required per cubic inch to produce a stress high enough in the coal specimen (a 3-in. cube) to cause failure. These values were obtained by extrapolating the stress-strain curve to failure and may involve some error. For curve *B*, the energy required to produce a stress in the coal of ½ the ultimate stress was measured. It is interesting that in this curve the results show considerably more uniformity than in curve *A*. These two curves indicate, as would be expected, that the energy a coal specimen can store increases as the ultimate stress increases.

It would be of interest to know how much of the energy stored in a coal specimen may be released when the coal specimen fails because of overstress. It is impossible to answer this question with the data at hand. The experimental results of Lawall and Holland show that coal when stressed to

90 per cent of its ultimate strength may release 85 per cent of the energy stored in the coal specimens when the stress is released, the remaining 15 per cent acting to produce permanent strains and heat in the coal specimen. How much of this energy is released when the specimen ruptures is unknown. The violence of the rupture of the stronger coals (coals having an ultimate strength above 3000 lb. per sq. in. in a 3-in. cube) suggests that a large portion of the stored energy is released.

The figures given in the preceding paragraph were obtained when the stresses in the coal existed 20 to 30 min. It is quite possible that coal stressed near its ultimate strength for a long time would have more of its stored energy used in producing permanent strain than these results would indicate. Greenwald found that for the comparatively low stress produced by the weight of the overburden the coal tested in small pillars was able to release energy that apparently had been stored during past geologic ages (p. 16 of ref. 10). Also, observations by Winstanley,[15] McCall,[16] Greenwald,[17] Maize,[18] and Holland (pp. 342 and 343 of ref. 14) suggest that coal pillars after being slowly loaded and compressed by force induced by mining subsequently may be unloaded and decompressed, thereby releasing part of the energy stored.

Some Properties of Coal-measure Rocks

Fourteen specimens of rock from the middle Pottsville measures in Fayette County, West Virginia, were tested to get an idea of the structural characteristics and their variability in a particular mine roof (Table 5). These specimens were all 2¼-in. drill cores from the same prospect hole. These results indicate:

1. The crushing strength of the rocks varied from 1795 to 15,930 lb. per sq. in. It is true that in these results there was considerable variation in the ratio of the length of the specimen to its diameter, which may have caused some of the shorter specimens to develop a higher crushing strength than they would have developed if they had been longer. The results, however, do indicate a

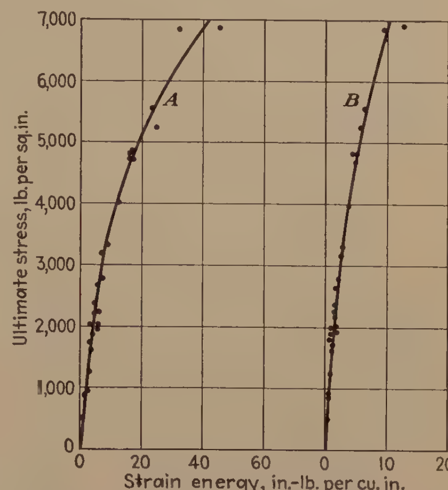

Fig. 12.—Strain energy stored by coal. Each point is the average of determinations made on two to four specimens. Strain energy values for stresses less than 1000 lb. per sq. in. calculated from Greenwald's experiments on small mine pillars.

wide variation in the crushing strength of the various rocks making up the mine roofs in this locality.

2. The stress-strain and elastic properties varied over a wide range. For example, the secant modulus of elasticity, the secant being drawn between 0 and 3000 lb. per sq. in., varied from a low of 980,000 to a high of 5,500,000 lb. per sq. in.*

3. The rocks tested were able to store from 1.3 to 7.9 in-lb. of energy per cubic inch at a stress of one-half the ultimate strength and were estimated to store as high as 22.8 in-lb. per cubic inch at the point of failure.

4. The rocks were able to release a large part of the energy stored when the stress was reduced to zero. Specimen 2, for example, after being stressed to the breaking point, released 44.4 per cent of the stored

* Specimen 2 excepted because it did not develop a stress of 3000 lb. per sq. inch.

TABLE 5.—*Physical Properties of Coal-measure Rocks*

Rock	Length, In.	Diameter, In.	Ratio of Diameter to Length	Crushing Strength, Lb. per Sq. In.	Secant Modulus of Elasticity,[a] Lb. per Sq. In.	Strain Energy, In-lb. per Cu. In., Required to Produce a Stress Equal to		Strain Energy Absorbed and Released[d]	Ratio of Energy Released to Energy Stored, Per Cent	Manner of Failure
						One-half Ultimate Strength	Ultimate Strength			
1. White sandstone with pebbles of slate. Fine grain except for pebbles	4¼	2 7/32	0.52	6,360	1,000,000	3.8	9.3	5020, 7.2, 3.9	54.2	Steep wedge
2. Coarse-grained porous sandstone, brown with darker brown streaks	6	2 1/16	0.34	1,795	230,000[b]	1.3	5.4	1795, 5.4, 2.4	44.4	Steep wedge
3. Banded gray shale (sandy)	4¼	2¼	0.53	10,060	980,000	6.7	17.6[c]	5020, 6.7, 3.8	56.7	Somewhat wedge-shaped but wedge not pronounced
4. Fine-grained white sandstone	6⅝	2¼	0.34	12,450	2,300,000	6.2	19.4[c]	5000, 4.6, 3.1	57.5	Failure very close to vertical shear. Slight tendency toward cone failure. Hard pebble in middle of specimen
5. Gray curly shale (sandy)	8	2¼	0.28	8,060	3,300,000	2.5	8.2[c]	5020, 3.3, 2.8	84.8	Pyramid
6. White fine-grained sandstone	9½	2¼	0.24	15,930	2,600,000	7.9	22.8[c]	6030, 4.7, 3.7	78.7	Vertical shear and wedge combined
7. White fine-grained sandstone	9⅝	2¼	0.23	7,040	3,150,000	1.8	4.9	7040, 4.9		Vertical shear and wedge combined
8. White fine-grained sandstone	11	2¼	0.20	11,280	2,050,000	5.6	18.3[c]	5020, 4.6, 3.4	76.9	Vertical shear with small cone at top
9. White sandstone with brown pebbles	5	2¼	0.45	10,000	1,180,000	5.9	16.7[c]	6281, 8.5, 5.5	64.8	Wedge
10. Brown fine-grained sandstone; crack showing on one side.	7¼	2¼	0.31	7,400	2,250,000	2.5	5.7[c]	5520, 4.4, 3.3	75.0	Failure along cracks. Combined vertical shear and wedge
11. Brown fine-grained medium-dense sandstone	3¼	2 5/32	0.66	12,700	5,500,000	2.9	5.0[c]	8780, 2.7, 2.0	74.2	Vertical shear with a small wedge in center of specimen on bottom
12. Brown streaked sandstone; unopened crack on one side.	3⅝	2¼	0.62	6,880	3,825,000	1.6	5.8[c]	6540, 5.1, 2.6	51.0	Wedge and vertical shear
13. Banded gray shale (sandy)	2¼	2¼	1.00	7,750	980,000	4.0	11.0[c]	5520, 7.6, 3.7	48.7	Vertical shear combined with horizontal and diagonal shear along bedding plane
14. Brown sandstone	7½	2¼	0.30	10,000	4,200,000	2.7	8.3	10100, 8.3, 8.0	96.2	Vertical shear and wedge

[a] Secant is drawn between 0 and a point on the stress-strain curve corresponding to 3000 lb. per sq. inch.
[b] Secant drawn between 0 and 900 lb. per sq. inch.
[c] Estimated by extrapolating the stress-strain curve to point of failure.
[d] The first figure given is a stress value, the second figure is the energy stored at that value of stress and the third figure is the energy released when the stress is reduced to zero. For example, when specimen 1 was stressed to 5020 lb. per sq. in., it absorbed 7.2 in-lb. of energy per cubic inch of specimen. When the stress is reduced to zero, 3.9 in-lb. of energy per cubic inch of specimen was released.

energy when the stress was reduced to zero and specimen 14, under the same conditions, released 96.2 per cent of the stored energy.

The stresses to which these rocks were subjected, of course, lasted for a very short time. It is quite possible that had the stresses lasted longer a larger deformation would have resulted and probably less of the energy stored would have been released when the stress acting was reduced to zero.

The stress-strain characteristics of the rocks tested are illustrated in Fig. 13 and the variation in the elastic properties under load in Fig. 14. It is worth noting that in every specimen except No. 2 the elastic modulus increased as the stress in the specimen increased. Sandstones and shales behave in this respect as do other rocks.[19] Figs. 15 and 16 are photographs of some of the rock specimens before and after failure. The manners of failure are rather irregular; it is rather striking that so many of the specimens failed mainly by vertical shear. Even when pure wedge failure did occur, the angles were very flat. Specimen 8 is especially noteworthy. The photograph shows this specimen split or sheared in two pieces. The faces formed are parallel with the direction of force application. The small cone formed indicates some tendency toward wedge failure. The crack extending downward from the bottom of the cone was not evident until two or three days after the specimen had been failed.

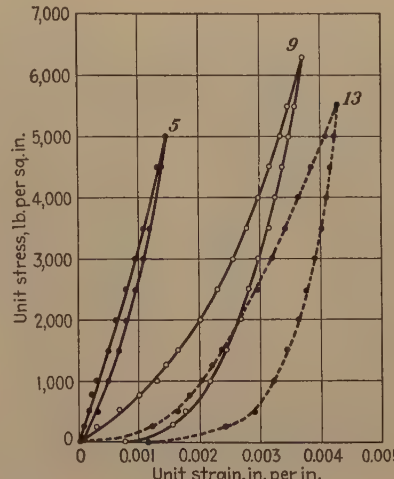

FIG. 13.—STRESS-STRAIN RELATIONSHIP OF VARIOUS ROCKS FROM MIDDLE POTTSVILLE COAL MEASURES.

Numbers on curves refer to specimen numbers in Table 5.

INFLUENCE OF PHYSICAL PROPERTIES UPON BUMPS

Information from the experiments upon thin specimens, although meager, seem to indicate that a coal pillar will not fall violently through the application of a uniform, slowly applied load if the ratio

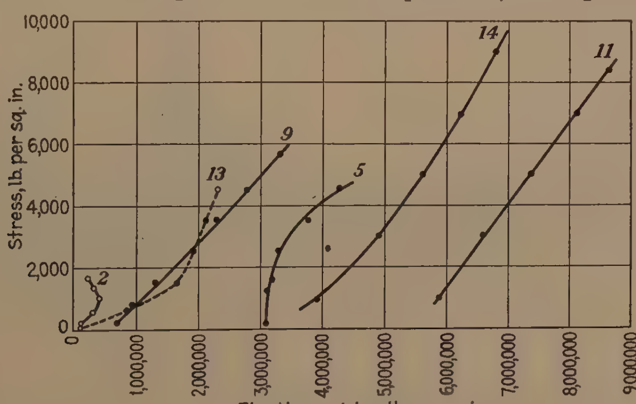

FIG. 14.—VARIATIONS OF ELASTIC MODULUS WITH INCREASING STRESS IN ROCKS.
Numbers on curves refer to specimen numbers in Table 5.

of the least width to the thickness of the pillar is sufficiently great. It also seems to be indicated that this ratio must be higher for strong coal than for weak coal, in order to prevent abrupt failure. This fact suggests that when deep seams are being mined, either of strong coal or weak coal, large pillars will be more effective in preventing bumps than the same area of support left in small pillars.

FIG. 15.—SOME ROCK SPECIMENS TESTED.
Left to right: brown sandstone, gray sandstone, banded sandy shale, white sandstone with brown pebbles, end view of a specimen, brown porous sandstone, gray curly shale.

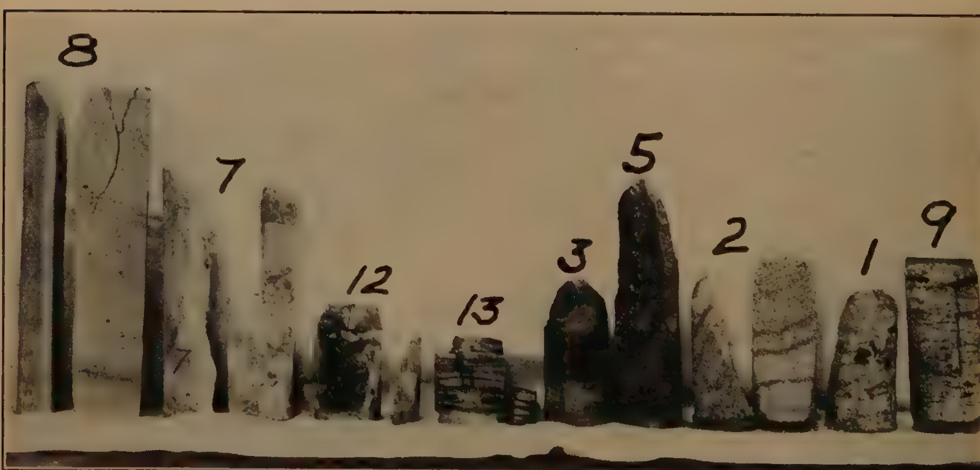

FIG. 16.—ROCK SPECIMENS AFTER FAILURE.
Numbers refer to specimen number in Table 5.

It should be borne in mind, however, that, owing to the variation in the stress-strain characteristics of coal situated short distances apart in a pillar, even if the roof did act to impose a uniform load the pillar would not necessarily have a uniform stress distribution. It seems probable that sections having a high elastic modulus would tend to carry a large part of the load. This means that in a given pillar

high stress areas may occur. If these areas are close to a free face of the pillar, the area may fail abruptly. This type of bump may occur in a large or a small pillar.

The fact that coal under compression stores elastic strain energy indicates that a large amount of the energy that causes the undesirable effects of the bump may come from the coal. The fact that strong coal stores greater amounts of energy before failure than weak coal suggests the reason that a strong coal seems to be one of the necessary factors in the occurrence of bumps. A weak coal may fail abruptly but as it does not store so much elastic energy, its failure is not nearly so disruptive and the particles are not thrown from the face nearly so violently.

The possibility of differential lateral expansion between the individual beds in a coal bed suggests that this property may cause shearing stresses to develop between the beds. The differential lateral expansion also may act to set up tensile stresses in the members having the smaller lateral extension characteristics. If this is true, pillar disintegration or spalling would possibly proceed at a higher rate than if the lateral extension characteristics in all members were the same. This action probably would tend to make large pillars less resistant to bumps.

The type of bumps occurring from these causes has been termed a pressure bump.[22] The physical properties of the coal-bearing strata as well as the properties of the coal bed itself enter into the shock or district bump. The factors that must be considered in this case are: (1) variation in the strength of the rocks composing the mine roof, (2) variation in the elastic properties of the rock composing the roof and floor, (3) the energy stored by the rocks composing the roof and floor, (4) the elastic strength and energy-storing properties of the coal bed itself.

The stress occurring in a particular bed having a sizable thickness in a mine roof acting either as a plate or an arch will depend upon its position in the roof and its elastic properties. A member having a higher elastic modulus than its adjoining members in a plate or beam will tend to have high stresses. If it occurs in an arch, it will carry more than its share of the total load and high stresses will develop. When these stresses become equal to the ultimate strength of the particular member, failure will occur. When failure occurs, the energy stored in this member will be released, and although the material of the member may have small amounts of energy stored per cubic foot, as the member may be thick, large quantities of energy will be released. When the coal bed is the weakest member in the strata in which this energy is released, it will absorb and serve as the relief valve, so to speak, of the energy released. Consequently, coal pillars that are heavily laden and already have absorbed large quantities of energy will not be able to meet the additional demands of mining, and abrupt failure is likely to occur. This is especially true of pillars having a low ratio of lateral dimension to thickness, or having high stress areas due to variation in the elastic properties of the coal bed. In connection with this action it must be remembered that a pillar having a large $\frac{l}{h}$ ratio may fail under impact even if it would withstand the same peak pressure applied gradually. It must also be kept in mind that, since coal and associated rock behave elastically, the stress wave induced by the failure of a strong roof member can be transmitted over considerable distance through the rocks themselves before it will be damped out. Therefore, the effects of the elastic failure of a roof member may be manifested a considerable distance from the location of the failure, or it may cause bumps over large areas almost simultaneously. It also seems possible that, owing to the sudden application of stresses of this kind, the coal may develop higher stresses

before failure than it would under gradually increased stresses. Consequently more of the energy released by the failed roof member will be stored by the coal, and so will be available at the instant of failure to cause very violent expulsion of the pieces from the failed pillar.

Prevention of Bumps in View of Physical Properties of Coal and Associated Rocks

The physical properties outlined in the preceding pages suggest that to prevent or minimize bumps the following conditions of mining should be followed:

1. The larger the pillar, the less the chance of abrupt failure, even though the total area of small pillars may be the same as the area in the large pillar.

2. Pillars should have larger lateral dimensions for a given thickness in strong coal than in weak coal.

3. Large pillars will tend to cut down bumps due to variations of elastic properties making up the pillar, because the amount of free face per unit of pillar volume in which blowouts can occur will be less than in small pillars.

4. Longwall mining using heavy pack walls to prevent high roof stresses and consequent abrupt failure is the best way in which to prevent shock bumps. Of the two methods of longwall mining, advancing longwall is preferable to retreating longwall, because in advancing longwall passages are not driven in the coal bed. This difference means that in advancing longwall the free faces in which "blowouts" due to high local stress can occur will be reduced to a minimum.

Predicting Bumps

In view of the data in the foregoing pages, it does not seem possible to predict the occurrence of bumps. Methods do not exist for locating areas having high elastic constants in a pillar or determining when a particular member or members in a mine roof are being overstressed. The most that can be said is that in areas having strong coals and thick strong sandstones, limestones, or shales close to the coal bed in both roof and floor, the mining system should be designed with the possibility of bumps in mind.

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References

1. G. S. Rice: Bumps in Coal Mines. *Trans.* A.I.M.E. (1936) **119**, 13.
2. O. Müller: An Investigation of Carbon Rocks as an Explanation of Pressures in Mines. *Glückauf* (Nov. 22, 1930) 1607.
3. H. Louis: Crushing Strength of British Coals. *Trans.* Inst. Min. Engrs. (1904–1905) **28**, 319.
4. E. Penman: The Crushing Strength of Coal Pillars. *Coll. Guardian* (1931) **143**, 700.
5. J. Daniels and L. D. Moore: The Ultimate Crushing Strength of Coal. *Eng. and Min. Jnl.* (1907) **84**, 263.
6. W. Griffith and E. T. Conner: Mining Conditions under the City of Scranton. U.S. Bur. Mines *Bull.* 25 (1912) 82–89.
7. A. N. Talbott: Compression Tests of Illinois Coal. Ill. State Geol. Survey, *Bull.* 4 (1907) 198.
8. G. S. Rice and C. Enzian: Tests of Strength of Roof Supports Used in Anthracite Mines of Pennsylvania. U.S. Bur. Mines *Bull.* 303 (1929) 5–7.

9. C. E. Lawall and C. T. Holland: Some Physical Characteristics of West Virginia Coals. Eng. Expt. Sta., West Virginia Univ., *Research Bull.* 17 (1937) 27.
10. Greenwald, Howarth and Hartmann: Experiments on Strength of Small Pillars of Coal in the Pittsburgh Coal Bed. U.S. Bur. Mines *Tech. Paper* 605 (1939) 12.
11. D. Bunting: Chamber Pillars in Deep Anthracite Mines. *Trans.* A.I.M.E. (1911) **13**, 198.
12. D. T. Griggs: Deformation of Rocks under High Confining Pressure. *Jnl. of Geol.* (1936) **44**, 550.
13. Greenwald, Avin and Rice: Compressibility and Bearing Strength of Coal in Place; Tests of Lateral Compression of Pittsburgh Coal Bed. U.S. Bur. Mines *Tech. Paper* 527 (1932).
14. C. T. Holland: Pillar Deformations in a Bituminous Coal Mine. *Trans.* A.I.M.E. (1938) **130**, 340.
15. A. Winstanley: Longwall Roof Control. *Coll. Guardian* (1931) **142**, 746.
16. T. L. McCall: Further Notes on Bumps in No. 2 Mine, Springhill, Nova Scotia. *Trans.* A.I.M.E. (1934) **108**, 61.
17. H. P. Greenwald and G. S. Rice: Studies of Roof Movements in Coal Mines. U.S. Bur. Mines *R. I.* 3355 (1937) 13.
18. E. R. Maize, Ed. Thomas and H. P. Greenwald: Studies of Roof Movements in Coal Mines. U.S. Bur. Mines *R. I.* 3506 (1940) 5, Fig. 6.
19. W. A. Zisman: Young's Modulus and Poisson's Ratio with Reference to Geophysical Applications. *Proc.* Nat. Acad. Sci. (1933) **19**, 660, 661.

DISCUSSION

(*Charles F. Jackson presiding*)

G. S. Rice,* Alexandria, Va.—The paper by Mr. Holland contains many points of interest for mining men. The subject was one with which I had to deal both in private practice and while I was in charge of mining methods in the U. S. Bureau of Mines.

I investigated many bumps, large and small, between 1916 and 1938, and made recommendations thereon in specific fields, which in many cases led to changes in local mining methods in order to prevent further bumps. The description of some of these are cited in papers given to various technical societies under the title of "Bumps in Coal Mines." A paper on this subject was published in the A.I.M.E. Transactions for 1936[1] and was followed by a paper by J. F. Bryson.[20] These studies were conducted in the Cumberland coal fields of Kentucky and Virginia, where there is a very heavy cover of strong rock. The recommended method of mining to meet the conditions was adopted with favorable results.

I note with interest that Mr. Holland apparently has accepted only in part my distinction between pressure bumps, which I have used to indicate rupture of a pillar that is overloaded, and "shock bumps," due to rupture of an overlying strong rock bed such as a thick sandstone acting as a beam or arch, where an unsuitable mining method leads to violent effects of strong shock waves, sometimes effecting distant disturbances with crushing of pillar and roof supports.

The remedy that has been found effective both in coal mines and in metal mines, such as the Lake Superior copper mines and the South African gold mines, has been the adoption of some form of longwall, which, by breaking the roof at short intervals, avoids the rupture of great spans of strong, rigid rocks in the roof measures.

I agree in general with Mr. Holland's conclusions, but he does not explain his method of extraction of the pillars. He advocates advancing longwall, and I have frequently used this method where it was feasible to do so, but not for pillar extraction.

His statement that it is not possible to predict the occurrence of bumps may be true for the first experience, but subsequently I think that they are usually predictable, and it is also possible to employ earthquake-recording apparatus of a portable nature, although this device has failed when used on the surface above the mines.

* Formerly Chief Mining Engineer of the U. S. Bureau of Mines (from its inception in 1908 until his retirement in October 1937).

[20] J. F. Bryson: Method of Eliminating Coal Bumps or Minimizing Their Effects. *Trans.* A.I.M.E. (1936) **119**, 40.

L. Obert,* Washington, D. C.—In the concluding paragraph Mr. Holland says that there is no method for measuring the modulus of elasticity of coal pillars in a mine. Bureau of Mines *Report of Investigations* No. 3444, "Measurement of Pressures on Rock Pillars in Underground Mines, Part I," describes a sonic method for measuring Young's modulus, and No. 3521, "Measurement of Pressures on Rock Pillars in Underground Mines, Part II," gives the results of measurements of Young's modulus made on rock and concrete (replacement) mine pillars *in situ*. Moreover, the second report describes a method worked out by the St. Joseph Lead Co. in conjunction with the Bureau of Mines, whereby the pressure exerted by the back can be measured. Both of these methods could easily be applied to coal mining and would serve to give the information lacking in Mr. Holland's report.

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C. T. Holland (author's reply).—Dr. Rice has investigated many bumps in coal mines as well as rock bursts in metal mines. The observation he has made and the theories he has formulated have been of great value to those engaged in the mining industry in areas where phenomena of this nature occur. In writing the paper under discussion, I did not attempt to go very far into the theories of causes other than to regard the bump in the final analysis as being caused by overstress. It seems entirely probable that the overstress may be caused in the two general ways Dr. Rice had advanced, pressure and shock. As I see the problem, however, we should remember that apart from the factors already discussed in my paper the pressure bump may result from a number of causes such as stresses built up by past geologic action, weight of the superincumbent strata, concentration of stresses in certain areas caused by mining operations, or stress concentrations caused by a highly mountainous surface, as well as many others. The shock bump may result from a momentary overstress caused by the rupture of a roof or floor stratum, failure of a pillar at some other locality, or the subsidence of overlying strata through a void created by mining and possibly other actions.

It is with considerable satisfaction that I note that Dr. Rice agrees with the suggestions made in the paper for the prevention of bumps. However, I wish to make it clear that the preventatives suggested are not believed to be absolute, but rather are believed to be the best procedure to follow if one is to attempt to prevent bumps. For example, advancing longwall has been used to a considerable extent in mining under heavy cover and bumps apparently have not occurred frequently when this mining method has been used. The experience at Pendleton colliery[21] in England, however, demonstrates that severe bumps may occur, especially when faults are present, even during the use of advancing longwall with pack walls. While at Springhill, Nova Scotia (ref. 16, p. 41), the experience to 1934 indicates that retreating longwall has not been effective in preventing bumps under the conditions prevailing. I also understand that retreating longwall has not been completely effective at Coal Creek colliery, British Columbia,* although it is believed to have mitigated the effects of the bumps that occurred. On the other hand, in the Harlan field, Kentucky, a retreating long face with the use of cribs has been effective in preventing bumps.[20] It seems quite possible under certain circumstances, especially where tectonic forces from past geologic disturbances are the causative agent, that bumps may not be preventable. In cases of this kind the engineer must direct his efforts toward lessening the severity of the bumps and their effects and protecting the mine workmen.

I am still of the opinion that bumps cannot be predicted with apparatus presently available, because in the final analysis a bump results from overstress of a pillar, or of a roof or floor stratum. As far as I am aware, proved practicable methods do not exist for indicating or detecting such an overstress. Portable earthquake-recording apparatus undoubtedly will register the vibrations resulting from the occurrence of bumps but it is questionable whether detectable vibrations are set up by the accumulations or concentrations of stress in the strata of mine pillars, roofs and floors. Until stress concentrations can be detected, bump prediction in general seems likely to be unsatisfactory. Dr. Rice states that after they are first experienced, bumps are usually predictable. I believe this statement to apply in the sense that unless a change in mining method or of some other nature is made bumps may be expected to continue to occur. However, the experience of mines having bumps apparently does not indicate that the prediction of bumps, as regards time and place of occurrence as well as severity, is generally successful.

Mr. Obert seems to have mistaken the meaning of the last paragraph in my paper. The statement is that methods do not exist for locating areas having high elastic constants in a pillar or for determining when a particular member or members in a mine roof are being overstressed. Mr. Obert's technique for finding the elastic modulus of a mine pillar seems to be readily applicable to coal-mine pillars, but it also appears to me that his method will apply in a practicable sense to finding the average modulus of elasticity of the entire pillar. It would be possible, I imagine, by using his method, and by doing a considerable amount of

[21] Crumps at Pendleton Colliery. *Coll. Guardian* (1927) 135, 533.

* Communication from B. Caufield under date of April 20, 1942.

drilling, to investigate the variations of elastic modulus in the sections of a large pillar, but it appears that the method would probably be so slow and costly as to render it impracticable unless improvements that cannot now be foreseen can be made. However, I do congratulate Mr. Obert upon the ingeniousness of his method and the resourcefulness he has shown in applying it. The problem he has attempted of measuring the stress in mine pillars is one that stirs the imagination of the mining engineer, but at the same time, as the author has occasion to know, it is a problem beset with considerable difficulty. If Mr. Obert can develop the method to the point where he can measure the average stress in the pillar, he probably will have contributed greatly toward perfecting a technique that can be employed in preventing the pressure bump and in investigating the general problem of roof, pillar and floor stresses in mining as well as the causes of creeps, squeezes and various other actions.

Occurrence of Bony Coal in Castle Gate D Seam and Its Effect on Ash-slagging Characteristics

By Claude P. Heiner,* Member A.I.M.E., and Carl S. Westerberg†

(New York Meeting, February 1941)

Observation of the clinkering action of coal from the Castle Gate D seam in underfeed stokers over a period of years has given rise to the present investigation of the effect of bony coal on clinkering. Some very costly examples of slagging clinkers in particular cases have been of especial interest to the authors. In such cases coal analyses and ash-softening and fluid temperatures have been found of little value. In general, it was noted that slagging was accompanied by a somewhat higher ash content of the stoker slack, and the suspicion naturally arose that foreign matter was the troublemaker. One would naturally suspect bony coal and rock impurities because of their occurrence and prevalence within the seam and the relative absence of sulphur or other impurities in the coal mined. The test results given herein indicate that the concentration of the bony coal and rock impurities in the coal determines the clinkering tendencies of the ash during combustion.

Castle Gate D Seam

Extending from the Utah-Colorado line near Grand Junction, Colo., westward to Greenriver, Utah, and thence in semicircular form north and westward to Castle Gate, and from there southward to Mt. Hilgard in central Utah, is a line of nearly perpendicular cliffs, composed of interbedded massive sandstone and shales, making up the coal-bearing horizon of the Mesaverde formation of Upper Cretaceous age. The cliffs face roughly concentrically toward the south and southeast and the strata, with conformably enclosed coal beds, gently dip away from the face of the cliffs and at right angles to their front line. Back of these cliffs the mountain masses rise to an altitude of from 8000 to 10,000 ft. above sea level. Stretching away from their foot is a great shale plain, on which the towns are situated and over which the railroad lines and highways run. The Castle Gate mines enter the coal beds at their outcrop; the workings extending within a particular coal bed away from the outcrop and under the mountain mass back from the cliff line (Fig. 1).

Throughout the Mesaverde formation in the Castle Gate general area, there are from two to five workable coal beds. Usually not all of these are coincidentally of workable size and merchantable quality throughout any given large tract.

In the Castle Gate mine area, the base of the Mesaverde horizon and coal measures is marked by a massive, cliffmaking, gray sandstone bed 60 to 80 ft. thick. Locally this is referred to as the "Castle Gate sandstone floor." Frank R. Clark, in U. S. Geol. Survey *Bull.* 793, designates it as the "Aberdeen sandstone." At Castle Gate the various coal beds are designated by the letters *A*, *B*, *C*, etc., commencing with the lowest bed.

Fig. 2 is a graphic generalized vertical section, compiled from physical exposures

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in or near the Castle Gate mine. The subject matter of this paper refers particularly to the product from the D seam as shown on that section.

mine districts. It has not been found possible to closely predict their concentration, either in the bed itself or in the coal mined. Occurrence of bony coal within the

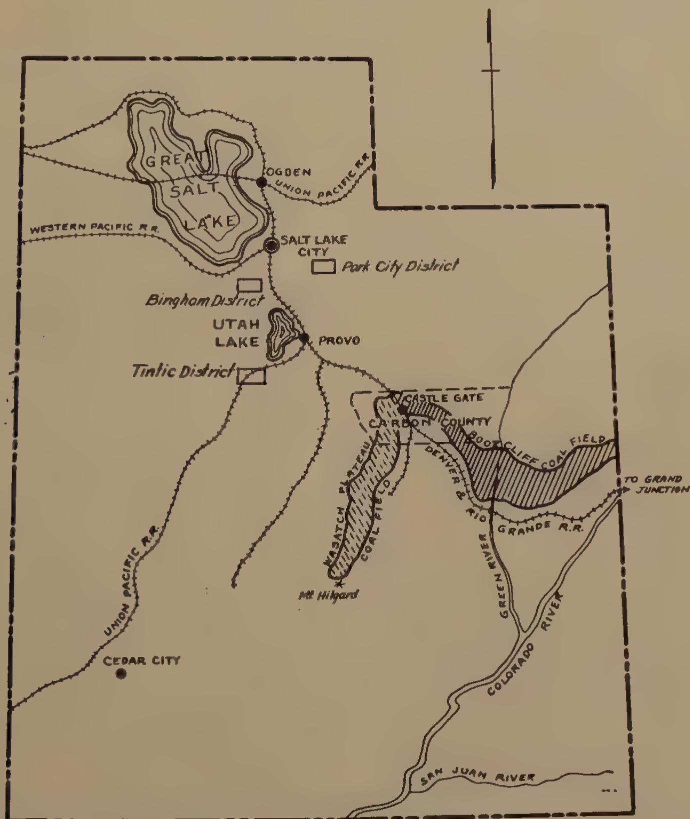

FIG. 1.—LOCATION OF PRINCIPAL COAL FIELDS AND METAL-MINING DISTRICTS IN THE STATE OF UTAH.

OCCURRENCE OF BONY COAL AND ROCK IMPURITIES

Impurities encountered in mining Castle Gate D seam include dense, irregularly banded layers and streaks of high-ash (20 to 35 per cent) coal generally called "bony," occasional rock "spars" or lenses, and splits, composed of sand rock, sandy shale or "bony" shale, also roof rock and foreign matter sometimes mechanically cut from floor. These impurities are generally localized in occurrence, and vary widely in concentration as between bed, also rock splits near the roof, is illustrated by Fig. 3. Sections *A*, *B* and *C* on Fig. 2 represent typical bed sections in major districts of Castle Gate No. 2 mine.

MINING AND PREPARATION METHODS

Mining practice consists of undercutting, shearing, drilling and blasting and mechanical loading. With this complete mechanization and thorough wetting down of the coal before loading, it is not possible to clean out the impurities before sending the coal to the preparation plant.

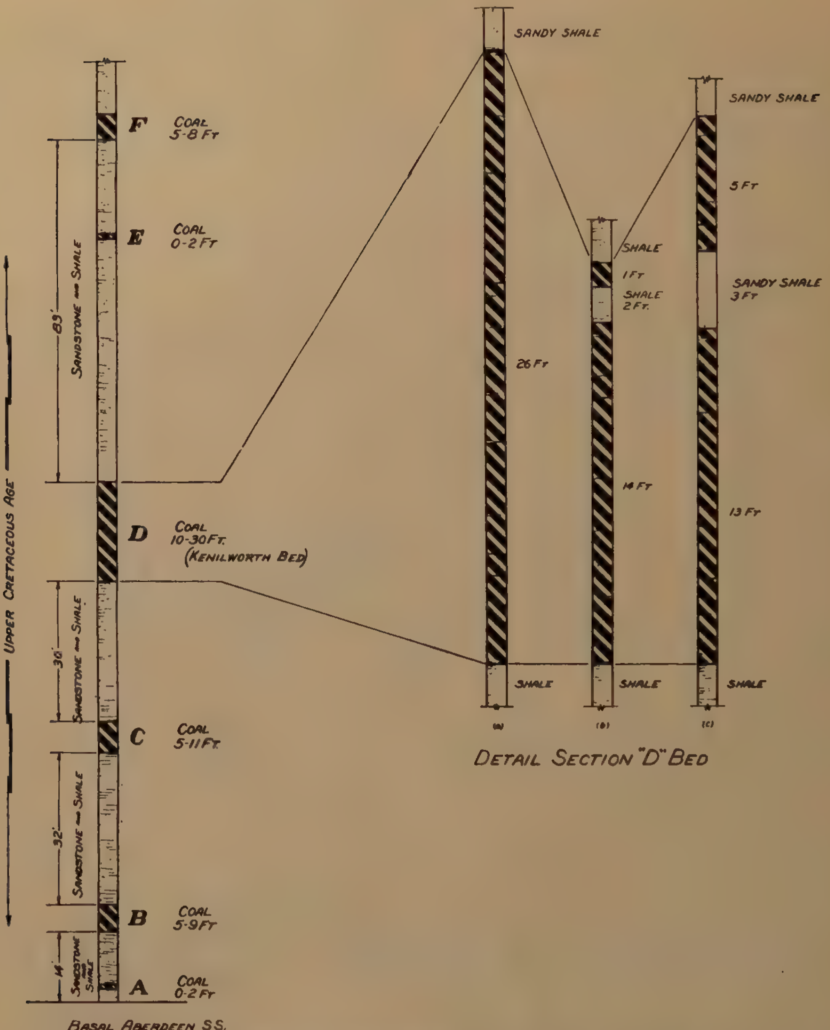

FIG. 2.—TYPICAL COLUMNAR SECTION THROUGH COAL MEASURES, CASTLE GATE NO. 2 MINE DISTRICT.

FIG. 3.—EXAMPLE OF COAL FACE IN CASTLE GATE D SEAM (CASTLE GATE NO. 2 MINE, UTAH FUEL CO.).

Read up:

Coal, temporary roof	
Shale, parting	1 in.
Sandy shale split	4 in.
Coal, with bony streaks	8 ft. 3 in.
Coal floor	

Coal thickness generally 4 to 6 ft. above parting shown. Bony streaks not continuous over wide areas.

FIG. 3.—FOR LEGEND SEE OPPOSITE PAGE.

With the present methods of room-and-pillar mining, first extraction takes 8 to 10 ft. of the bed above the floor, and where overburden and roof conditions permit the remaining top coal is shot down and loaded out on the retreat. Where rock splits and partings occur near the top of first extraction, caving to the roof (top rock) sometimes occurs.

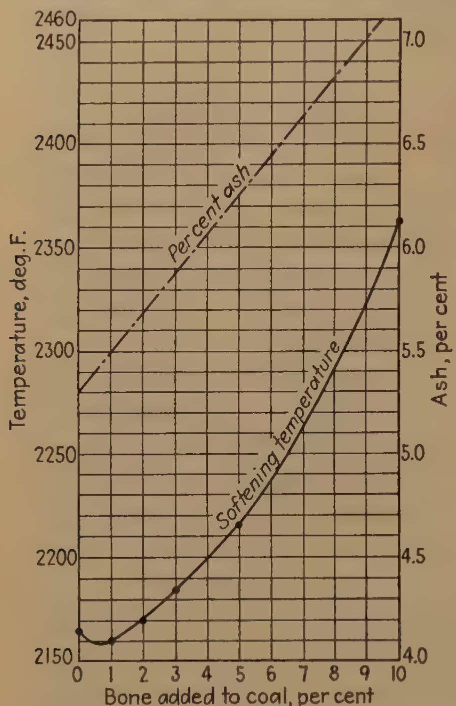

FIG. 4.—EFFECT OF ADDITIONS OF BONY COAL ON ASH-SOFTENING TEMPERATURES OF CLEAN COAL.

Present preparation practice includes hand picking of the larger sizes (plus 1⅝ in. and plus 3 in.) and either simple screening of the minus 1⅝-in. size or washing in a Baum type (Link-Belt Simon-Carves) jig of the minus 3-in. size. The latter process was used for the preparation of the clean coal and secondary refuse used in the stoker tests noted herein.

PRELIMINARY INVESTIGATIONS

Occasional complaints about slagging of clinkers from raw 1-in. to 0 slack coal and unreliability of softening and fusion temperatures as a guide to the source of the trouble led to the preparation of hand-picked samples to determine the effect of bony additions upon the cleaned coal. Samples of the bony coal were taken from the hand pickings (3 to 1⅝-in. and 8 to 3-in.) wasted as refuse from the raw-coal tipple and were added to the clean coal—also hand-picked—and the effect on softening temperatures with concentrations up to 10 per cent of the refuse were obtained (Fig. 4 and Table 1). Although a slight depression in softening point was noted when 1 per cent of the bony was added to the coal, no especial significance was attached to it. At first, it was thought that the increased softening temperatures resulting from additions of bone to coal would prevent clinkering to some extent.

TABLE 1.—*Effect of Bony-coal Additions on Ash-softening*

Temperatures of Clean Coal, Castle Gate D-seam Coal

Bone Added to Coal, Per Cent	Determined Ash-moisture-free, Per Cent	Softening Temperature, Deg. F.
0	5.3	2165
1	5.8	2160
2	5.9	2170
3	6.1	2184
5	6.6	2216
10	7.2	2361

However, the suspicion still existed that bony coal was harmful, therefore it was decided to repeat the experiment, determining also the fusion point of the ash. Additional hand-picked mixtures were made up, containing the same concentrations of bony coal, and both ash-softening and fusion points were determined (Fig. 5 and Table 2).

The results of this experiment gave an entirely different picture and pointed to the fact that additions of from 2 to 5 per cent of bony coal to the clean product had a very serious effect in depressing the fusion point of the resultant ash. Another similar experiment was conducted as a check, and

comparable results were obtained. Samples of each set of experimental data were taken from different sections of the Castle Gate No. 2 mine. In general, results with the different samples checked; that is, ash-softening temperatures followed the pattern of Figs. 4 and 5 and values for ash fusion followed Fig. 5. It was concluded therefore that additions of bony coal were responsible for the occasional slagging encountered.

At the same time, it was noted that beyond the limits of 2 to 5 per cent bony additions, both the ash-softening and fusion temperatures increased with the increasing concentrations of bony material, and it was found that there was no agreement between these laboratory determinations and the clinkering or slagging tendency; no reduction in slagging being noted with increasing fusion temperatures. In order further to clarify the laboratory results, ash analyses were made of the ash from the hand-picked coal and the bony coal, as given in Table 3. Included also in this table is an ash analysis on a clinker mass taken from a large underfeed stoker, which had slagging difficulty with this coal.

TABLE 2.—*Effect of Bony-coal Additions on Ash Fusion and Softening Temperatures of Clean Coal, Castle Gate No. 2 Mine, D Seam*

Sample No.	Bone, Per Cent	Ash Determined, Per Cent	Softening Temperature, Deg. F.	Fluid Temperature, Deg. F.
1 (clean coal)	0.0	4.1	2165	2368
2	0.5	4.2	2184	2368
3	1.0	4.2	2184	2368
4	2.0	4.3	2193	2276
5	3.0	4.5	2193	2260
6	4.0	4.7	2193	2260
7	5.0	4.9	2184	2308
8	6.0	5.0	2211	2308
9	7.0	5.2	2244	2308
10	8.0	5.3	2308	2308
11	9.0	5.4	2347	2347
12	10.0	5.9	2375	2375
13 (all bone)	100.0	21.1	Out of range	

The ash of the clean coal (column 1) is essentially acid in its high silica content, and the bone ash (column 2) is high in basic constituents, totaling over 80 per cent in mixed calcium and magnesium oxides. The analysis of the clinker mass (column 3) represents an objectionable mixture of

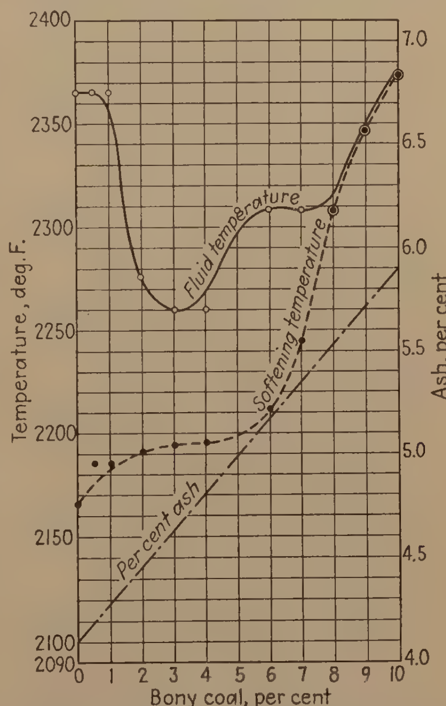

FIG. 5.—EFFECT OF ADDITIONS OF BONY COAL ON ASH FUSION AND SOFTENING TEMPERATURES OF CLEAN COAL.

coal ash and refuse ash. The percentage of iron oxide (Fe_2O_3) is higher than would be expected, probably owing to corrosion of the iron grates actually reported in this instance, by the fluxing action of the ash.

The relatively high silica content of the slag probably reflects more sand rock mixed with the bone coal than the bony ash analysis indicates. Obviously, from the high calcium content of the slag, considerable basic bony material has been added to the clean coal.

INVESTIGATIONS WITH WASH-BOX PRODUCTS

After the completion of a mechanical cleaning plant at Castle Gate early in 1940,

samples of wash-box float coal and secondary refuse were obtained in order to check more carefully the clinkering and slagging tendency with varying refuse concentrations. Preliminary investigations had shown the average specific gravity of bony material as about 1.685, and that sometimes it went as low as 1.610 and as high as 1.695. In the preparation of the samples, the wash box was operated at approximately 1.50 gravity. Visual examination of samples showed that the bone did not break clean from the coal, with the result that some material composed of part bone and part clean coal was necessarily included in the float portion of the wash-box product; no attempt was made to evaluate the extent of this banding of bony in the cleaned coal. Samples of 1-in. to 0 stoker coal were simply taken from the wash-box float discharge and were used as the basis of stoker tests, along with ground secondary refuse (3-in. to 0 crushed to minus ⅛-in. to 0). Float-sink separations at 1.50 gravity and ash determinations were made of the samples of wash-box float coal and secondary refuse, with results shown in Table 4. Ash analyses were also made of the 1.50 float portion (98.0 per cent) of the wash-box float coal and both the 1.50 float (48.1 per cent) and sink (51.9 per cent) portions of ground secondary refuse, as shown in Table 5.

Mixtures of wash-box float coal (5.9 per cent ash) and ground secondary refuse (23.8 per cent ash) were made up in 100-lb. lots to give increasing bony-coal concentrations as follows (see Table 6):

Bony Coal, Per Cent	Wash-box Float Coal, Per Cent	Bony Coal, Per Cent	Wash-box Float Coal, Per Cent
1	99	12.5	87.5
2	98	15	85
to	to	17.5	82.5
10	90	20	80

In addition to these tests, mixtures were made up to show the effect of hand-picked crushed (to minus ⅛-in. to 0) roof rock upon clinkering, the assumption having

TABLE 3.—*Ash Analyses, Clean Coal, Bony and Slag, Castle Gate Coal, D Seam*

	(1) Hand-picked Clean Coal, 1-in. to 0	(2) Hand-picked Bony Coal, 8-in. to 1⅝-in., Crushed	(3) Clinker (Slag) from Stoker
Percentage of ash	4.1	21.1	
Ash analyses, per cent:			
Silica (SiO_2)	48.59	3.16	44.81
Iron oxide (Fe_2O_3)	7.47	5.41	12.64
Alumina (Al_2O_3)	14.70	1.73	11.56
Calcium oxide (CaO)	13.74	63.93	24.35
Magnesium oxide (MgO)	2.48	18.64	
Alkali (K_2O)	4.71	5.13	6.64
Undetermined	8.31	2.00	
Total	100.00	100.00	100.00
Seam condition where sample originated	Streaked, bony, small amount rock	Same	Streaked, bony, and rock lenses

TABLE 4.—*Ash and Float-sink Determinations of Samples Used in Tests*

Sample	Head Sample		1.50 Float		1.50 Sink	
	Per Cent	Per Cent Ash	Per Cent	Per Cent Ash	Per Cent	Per Cent Ash
Wash-box float, 1-in. to 0	100.0	5.9	98.0	5.6	2.0	28.1
Secondary refuse, 3-in. to 0	100.0	23.8	24.1		75.9	
Secondary refuse crushed, minus ⅛-in.	100.0		48.1	7.0	51.9	34.3

been made that the very high silica content of such impurity would raise the ash-fusion temperatures enough to overcome slagging. Accordingly, repeat mixtures of 10 per cent roof rock with 90 per cent wash-box float coal were prepared and tested under firing conditions, as noted under Results Obtained.

This mixture caused severe slagging on repeat tests, therefore the assumption was made that concentrations of lime interspersed as binder in the sandy roof rock was the source of trouble. This was checked by noting active effervescence of the roof-rock samples in dilute hydrochloric acid solution. Additional samples containing 10 per cent sand rock were made up after testing each hand-picked rock cube in HCl

TABLE 5.—*Ash Analyses, Clean Coal and Refuse (Bony) Castle Gate D Seam, Castle Gate No. 2 Mine*

	1.50 Float 1-in. to 0 Coal	Crushed Secondary Refuse 3-in. to 0 Crushed to −1/8-in. to 0	
		1.50 Float	1.50 Sink
	(1)	(2)	(3)
Per cent ash	5.6	7.0	34.3
Per cent head sample	98.0	48.1	51.9
Ash softening temperature	2145°F.	2368°F.	2599°F.
Ash fluid temperature	2250	2417	2599
Ash analysis, per cent:			
Silica (SiO_2)	41.47	28.56	26.96
Iron oxide (Fe_2O_3)	7.01	6.38	7.02
Alumina (Al_2O_3)	16.43	9.50	6.14
Calcium oxide (CaO)	16.26	34.44	49.87
Magnesium oxide (MgO)	4.43	6.29	7.23
Alkali (K_2O)	2.88	4.27	0.14
Undetermined	11.52	10.56	2.64
Total	100.00 %	100.00 %	100.00 %
Seam condition where sample originated	Rock splits, spars, bony	Same as (1)	Same as (1)
Seam thickness where sample originated	8′, 14′, 20′	Same as (1)	Same as (1)

TABLE 6.—*Description of Mixtures Used and Stoker Results Obtained in Study of Ash Characteristics of Coal-refuse Combinations*

Sample No.	Washed Coal in Mixture, Per Cent	Secondary Refuse in Mixture, Per Cent	Equivalent in 1.50 Sink Bony as Fired (Calc.)	Ash Content, M. F.		Ash Fusion		Stoker Test Results		
				Determined, Per Cent	Calculated, Per Cent	Softening, Deg. F.	Fluid, Deg. F.	Pin Shearing	Chipped from Retort, Furnace	Number Times Fire-cleaned
	(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)	(9)	(10)
1	100	0	1.00	5.9		2180	2258	No		
2	0	100		23.8		2621	2621			
3	99	1	1.51	6.0	6.1	2155	2246	No		
4	98	2	2.02	6.4	6.3	2155	2246	No		
5	97	3	2.53	6.5	6.4	2155	2170	Yes		
6	96	4	3.04	6.6	6.6	2165	2184	Yes		
7	95	5	3.55	6.7	6.8	2175	2220	Yes		
8	94	6	4.06	7.1	7.0	2163	2248	Yes		
9	93	7	4.57	7.2	7.2	2211	2264	Yes		
10	92	8	5.08	7.4	7.3	2228	2287	Yes	Yes	
11	91	9	5.59	7.7	7.5	2232	2288	Yes		
12	90	10	6.10	7.8	7.7	2234	2310	Yes	Yes	
13	87.5	12.5	7.38	8.4	8.1	2245	2399	Yes		Yes 2
14	85	15	8.65	8.8	8.7	2260	2440	Yes		Yes 2
15	82.5	17.5	9.93	9.4	9.2	2278	2471	Yes		Yes 2
16	80	20	12.20	10.0	9.8	2290	2480	Yes		Yes 2
17	90	10 (roof rock)		11.2		2165	2284	Yes	Yes	Yes 2
18	90	10 (sand rock lime-free)		13.7		2280	2585	No	No	Yes 2

solution to obtain clean high-silica lime-free rock. The resultant mixture was run through the stoker, with the results shown in Table 6. (See also Results Obtained.) charted (Fig. 6). As noted above, ash and float-sink determinations were made on the crushed secondary refuse and total 1.50 sink material was calculated on the mixture

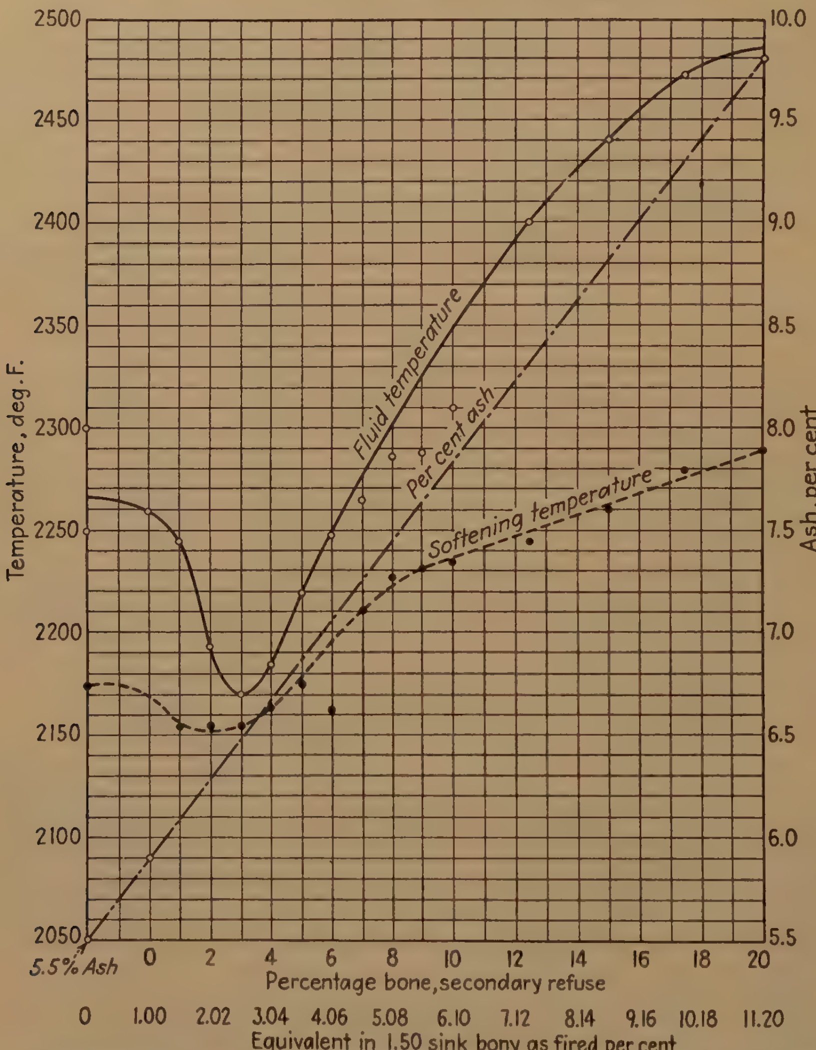

FIG. 6.—RELATIONSHIP OF BONY CONCENTRATIONS UPON ASH FUSION AND SOFTENING TEMPERATURE.

Each sample was analyzed for ash as a check against calculated ash on a moisture-free basis. Ash-softening and fluid temperatures also were run on each 100-lb. sample for the stoker tests, and the data (Table 6). In making these calculations, it was assumed the 2.0 per cent sink material in the wash-box float coal to be ground to minus ⅛ in. by crushing action in the stoker-feed tube, and that this portion of

the refuse had the same float-sink analysis as the crushed secondary refuse.

Screen analysis of the clean 1-in. to 0 coal was taken before and after it passed through the stoker worm, with the following results:

Size, In.	Coal in Hopper, Per Cent	Coal in Retort, Per Cent
1 to ½	31.8	9.6
½ to 3/16	35.4	40.1
3/16 to ⅛	2.1	5.4
⅛ to 0	30.7	44.9
	100.0	100.0

From the screen analysis of the coal in the retort it was evident that additions of ground ⅛-in. to 0 refuse mixed intimately with the clean coal, thus facilitating clinkering action of the mixture.

Stoker Tests

A small water-heating stoker and furnace were set up as shown in Fig. 7; detailed longitudinal section of stoker and furnace is shown by Fig. 8. The furnace was lined with refractory material, and the furnace top consisted of a cast-iron heat-absorption crown, through which was maintained a constant circulation of town water. Under continuous operation, the stoker feed rate was 5 lb. per hour, and since firing was controlled to burn 15 min. on, 5 min. off (Fig. 9), during the entire duration of each test, a total of 3.75 lb. of coal was burned per hour. Maximum available furnace-combustion volume above tuyere ring amounted to approximately 0.65 cu. ft. Each burning test, therefore, required approximately 26 hr. to complete. Maximum furnace temperature obtained during burning test of clean coal as estimated by optical pyrometer just prior to the end of "on" period was approximately 2150°F. No other furnace or fuel-bed temperatures were measured or thought necessary, because of uniform conditions as between tests. The furnace was connected to 2 ft. of 5-in. stovepipe leading to cyclone-dust and fly-ash collector, to which in turn was attached 18½ ft. of 5-in. pipe discharging to the atmosphere. Furnace and cyclone collector were cleaned out thoroughly after each test.

Fig. 7.—Water-heating stoker and furnace.

Results Obtained

1. *Explanation of Ash Analyses in Causing Slagging.*—In seeking to explain or point out a possible cause of slagging at the relatively low firebox temperatures obtained in the stoker tests, only one criterion is suggested—the chemical analysis of the ash in question. As mentioned previously, each analyzed ash is a very complex mixture, and probably would defy any attempt to determine its exact fusion characteristics on purely theoretical grounds. However, one factor stands out prominently: Whenever the concentration of calcium and magnesium oxides in the coal ash increases at the expense of the silica concentration, the result tends to an

unsatisfactory clinkering condition because of slagging. The source of this excess calcium and magnesium is largely in bone and rock impurities occurring in the bed of coal described. Control of the magnesium and calcium concentrations in the ash resolves itself into control of the amount of impurity shipped with the coal.

2. *Action of Cones during Ash-fusion Determinations.*—In the course of the investigations, it was learned that in obtaining softening and fluid temperatures the behavior of cones made from clean coal ash was different from that of cones made from coal-refuse ash. Coal-ash cones used in the fusion determinations made from clean coal retained their original size until the softening temperatures were reached. Coal-ash cones containing measured amounts of bone shrank visibly before the softening temperature was reached. When the cones were made from clean coal containing 10 per cent or more of bone, the cones shrank to less than one-half their original size before the softening temperature was reached (Fig. 10). The reason for this difference between the cones must lie in differential melting of constituents. The peculiar action of high-refuse cones may indicate probable inception of slagging by certain refuse constituents in actual firing tests.

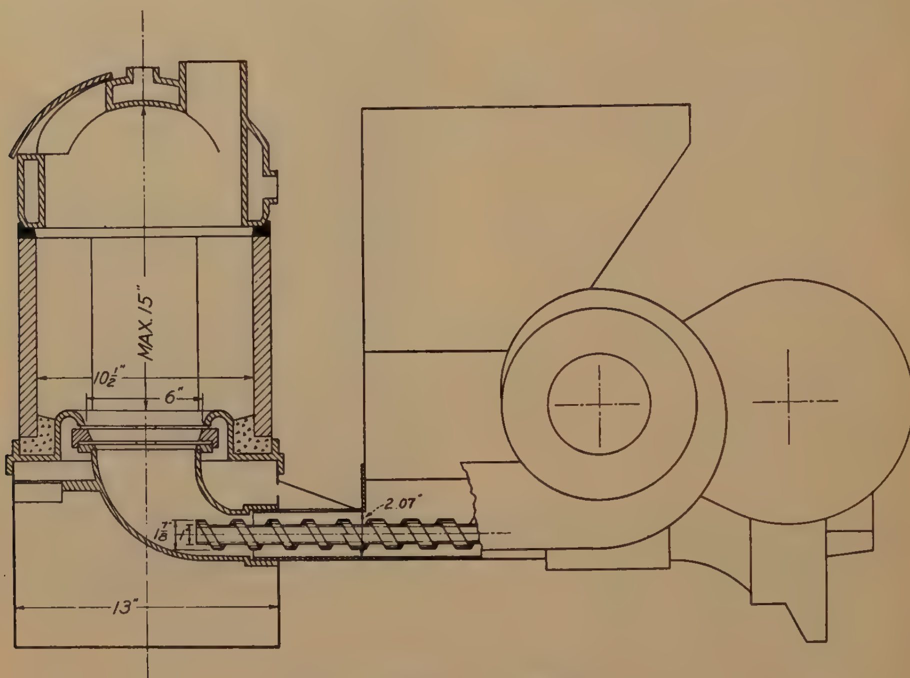

FIG. 8.—STOKER AND FURNACE.

3. *Origin of* CaO + MgO *in Float-sink Portions of Samples.*—Ash analyses of 1.50 float portion of wash-box float coal and float-sink portions of ground secondary refuse, as shown in Table 5, give evidence that the principal source of high lime content lies in the 1.50 sink portion of the ground secondary refuse with a small amount originating from the similar portion of 1.50 sink material from wash-box float coal. Using ash percentages as given in Table 5 and float-sink data from Table 4,

origin of ash and lime of a mixture of 10 per cent refuse and 90 per cent coal would be about as shown in Table 7.

possibility of a separation at 1.50 of crushed secondary refuse and the inclusion of this float portion with the 1-in. to 0

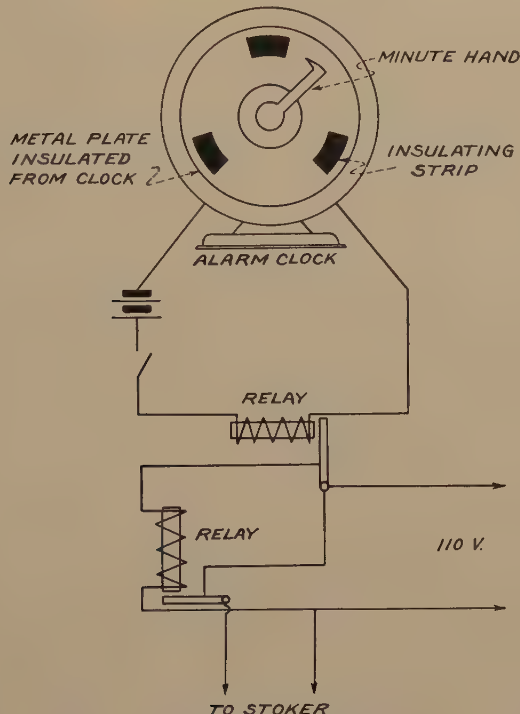

FIG. 9.—ARRANGEMENT USED TO OPERATE STOKER FOR PERIODS OF 15 MINUTES ON AND 15 MINUTES OFF.

This indicates that the float portions of ground (to minus ⅛-in.) refuse products (II and III) yield even less fluxing material

washed coal herein described. Such recleaned secondary refuse would cause less harm to the clinkering tendency than the

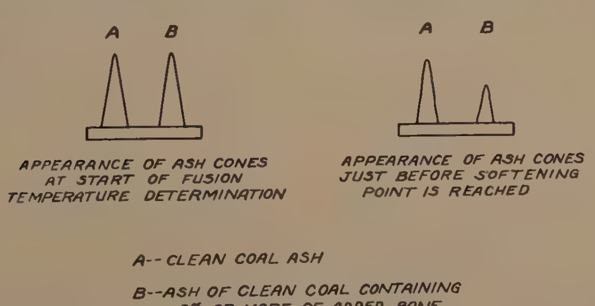

FIG. 10.—BEHAVIOR OF ASH CONES DURING FUSION DETERMINATIONS.

as CaO + MgO than the sink portion of the ground refuse material going over with the washed coal (IV). This would indicate

present inclusion of 1.50 sink material—amounting to 2.0 per cent—in the washed 1-in. to 0 coal. Further, approximately

eight times as much of the CaO + MgO originates from 1.50 sink portions (IV and V) of ground refuse as from the float portions (II and III). Recleaning of finely crushed refuse might therefore be justified as a means of coal recovery.

3 per cent secondary refuse—the point at which pin-shearing commenced.

Particular attention should be focused on samples 17 and 18, Table 6. Sample 17 consisted of a 10 per cent roof rock (from picking table) mixture with 90 per cent clean coal. This mixture caused severe slagging and the slag adhered to both the furnace refractory and retort castings. No picture was taken of this clinker sample, but for check purposes repeat tests were run with similar results.

TABLE 7.—*Origin of Ash and Lime in Refuse-coal Mixture*

Portion Studied	Ash Attributable to Portion Studied, Per Cent	CaO + MgO, Per Cent	CaO + MgO Attributable to Portion, Per Cent
I 1.50 float in wash-box float	4.93	20.7	1.02
II 1.50 float in ground wash-box sink	0.06	20.7	0.02
III 1.50 float in ground secondary refuse	0.34	40.7	0.14
Total 1.50 float coal, as burned	5.33	22.1	1.18
IV 1.50 sink in ground wash-box sink	0.32	58.1	0.19
V 1.50 sink in ground secondary refuse	1.79	58.1	1.06
Calculated percentages, this basis	7.44	32.7	2.43

4. *Character and Fluidity of Clinker Samples* (*See Fig.* 11).—Clinkers recovered from the test stoker were so arranged as to show the increase in black slag or glass with increased additions of bone coal to the clean coal. The black, glassy portions of each clinker were retouched with flat black paint so as to minimize highlights and photograph better. When the clinker had to be chipped from the retort (8 to 10 per cent bone) the pieces were badly broken, and the illustration is a poor example of the appearance of the clinker before this was done. This clinker had adhered on cooling to the refractory brick walls of the furnace. When pins were sheared (3 per cent to 20 per cent bone) the fluid slag had run into the tuyeres, there cooled and solidified, and stopped the feed of coal to the retort. Experience in this series of tests closely duplicates the trouble caused users of this coal in instances where slagging occurred.

It should be noted that character and fluidity of clinkers became progressively worse as increased additions of refuse were mixed with the clean coal. No letup in the slagging tendency or undesirability of clinkers was noted with increases beyond

Sample No. 18 (see also Fig. 11) consisted of 10 per cent sand rock, which had been carefully selected and tested with acid to remove basic constituents, and 90 per cent clean coal. The resultant clinker was porous and no evidence of slag existed, as clearly shown in Fig. 11. The obvious conclusion is that additions of silica (acid) to the clean coal will lessen the slagging tendency.

5. *Present Methods of Refuse Removal.*—Means of removing impurities referred to consist of: (1) hand picking of coarser sizes (now nominally +1⅝ in.), so as to remove bony material from any large coal that may later be crushed to stoker sizes and (2) gravity separation, by modern coal-washing processes, of all coal intended for stoker use. The problem of the best method to accomplish the desired cleanliness is dependent upon local conditions, such as concentration of impurities, available coal reserves

and tonnage requirements, as well as upon water supply and availability of cheap power.

latter being separable from the pure refuse after fine crushing.

Bony coal and rock impurities encoun-

FIG. 11.—CLINKER FROM MIXTURES OF CLEAN COAL AND SECONDARY REFUSE.

CONCLUSIONS

Slagging of clinkers when burning Castle Gate D-seam coal in underfeed stokers increases with increased concentrations of wash-box secondary refuse with the wash-box float coal (nominally 1.50 float) up to a point of 20 per cent concentration of refuse, the limit of these investigations. This secondary refuse is composed of mixed rock and "bony" coal with some clean coal, the tered in mining within the D seam are both slag-promoting in combination with the washed coal, because lime (basic) concentrations in the impurities produce fluxing action with the high-silica (acid) ash from the washed coal. The advantage of such removal of refuse from the stoker coal in reducing slagging tendency is in addition to the lessened amount of ash and higher fuel value of the cleaned coal.

While the tests were run only in one small underfeed stoker, the general application to other fuel-burning devices is clear. Separable (wash-box) impurities promote harmful and corrosive slagging of clinkers when intimately mixed and burned along with the cleaned coal, and their removal therefore is beneficial to the fuel for use in modern coal-burning devices.

Up to the limit of 20 per cent concentration of refuse, there seems to be little, if any, relationship between ash-softening (or fluid) temperature of coal-refuse mixtures and the slagging tendency. For example, a mixture of 10 per cent "bony" coal with 90 per cent wash-box float coal will produce more dangerous glassiness in underfeed stoker firing than the float coal itself, even though softening and fusion temperatures of the former are 50° to 80° higher than corresponding values for the float coal taken alone. Behavior in furnaces, therefore, is better determined by the separable refuse concentration than the so-called fusion point.

No attempt has been made to prove that the cleaned coal will not produce undesirable clinkers or slagging conditions where higher furnace temperatures are encountered or under burning conditions different from those of the tests run. However, it is felt that better and more predictable results will accrue from the removal from stoker sizes of bony and rock impurities encountered in mining.

Acknowledgment

Grateful acknowledgment is due to: Mr. E. H. Burdick, Mining Geologist in Salt Lake City, for his valuable suggestions and guidance in making up the generalized section of the coal measures at Castle Gate, Utah, and for his advice on other phases of the subject matter; also to members of the engineering staff of the Utah Fuel Co., including Mr. Fritz Nyman, Chief Engineer, and Wesley Hyatt, Mine Engineer, for helpful criticism of various data.

The authors also acknowledge the assistance of M. Elmer Christensen, Utah State Chemist, and E. F. Bailey, Superintendent, Jordan Plant, Utah Power and Light Co., in furnishing analyses and operating results with Utah coal in industrial furnaces.

Table Practice at the Mines of the Alabama By-Products Corporation

By H. J. Hager* and P. H. Haskell, Jr.,† Members A.I.M.E.

(New York Meeting, February 1941)

For the past 20 years Alabama has probably led all other coal-producing districts in the proportion of coal prepared by wet washing. All rail mines, with one exception, and a high percentage of truck mines now wash all coal under 3 or 4 in. Most of this coal has been cleaned by jigs, though within the past few years one or more wet-table plants have been installed for cleaning some of the coal from practically all the major seams. These tables supplement either single or multicell jigs usually for one of the following purposes: (1) to wash the finer sizes of raw coal, (2) to rewash the finer sizes of washed coal after they leave the jigs, (3) to re-treat jig middling products. The main purpose of this paper is to present cleaning data and operating factors involved in the preparation by tables of the finer sizes of raw coal. The specific information and data pertain to the Mary Lee seam of coal as extracted at three of the mines of the Alabama By-Products Corporation, which produces coal for both byproduct coke-oven consumption and the general steam market.

The Mary Lee seam has long been recognized by coal-preparation authorities as presenting a very difficult ash-reduction problem. Washing characteristics and physical structure of the seam vary widely. Its thickness ranges from 32 in. with no rock parting to 100 in. with from one to three partings. Over the entire field the bed contains bands of bone and rash. These are distributed throughout the seam cross section. A portion of this bone or bony coal occurs as definite bands 1 in. or more thick; however, some bone is interlocked as layers within the coal and consequently breaks into flakes when the coal is broken. This material is relatively high in ash and its specific gravity varies widely. It has been observed that in most cases in the washing of Mary Lee coal from 10 to 18 per cent of the raw coal is within ±0.10 specific gravity of the attempted point of separation. Therefore with the average wet jig, only with extreme difficulty can a low-ash product be obtained without loss of much marketable coal in the refuse.

Without entering into the controversial question as to whether modern washing equipment can efficiently handle simultaneously all sizes up to a given maximum, suffice it to state that the comparatively old-type jigs in service at the Alabama By-Products Corporation's mines could not clean the fine sizes satisfactorily. Therefore, in 1931 that company installed six diagonal-deck tables at the Barney mine, which were installed as an addition to a complete plant cleaning all coal below minus 3 in. The primary washed product is shipped as steam coal and another portion may be prepared as washed nut for domestic trade. These tables are washing 9 tons per hour of raw coal sized to ¼ in. to 0. The washed table coal is mixed with the 3 to ¼-in. jig product to constitute a 3 in. to 0 steam coal.

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Fig. 1 shows the cumulative ash curve on Barney 3 to 1½-in. and 5⁄16 in. to 0 raw coal as well as a table-products curve. A comparison of the two raw-coal curves

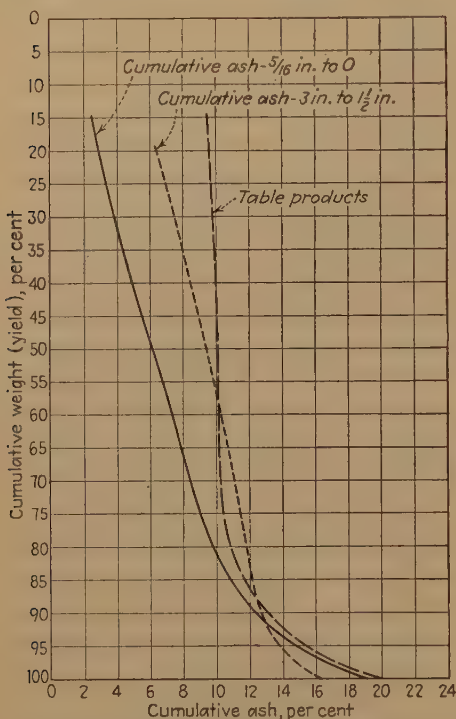

Fig. 1.—Cumulative ash curve on Barney 3 to 1½-inch and 5⁄16-inch to 0 raw coal; also a table-products curve, Barney mine, 5⁄16-inch to 0 coal.

shows that the ash will be reduced most effectively by efficient washing of the finer sizes. From float-and-sink data at 1.50 sp. gr., the 3-in. to 0 coal shows a cumulative ash of 12.2 per cent, whereas at the same gravity the 5⁄16-in. to 0 coal contains 10.4 per cent ash (Table 1). This characteristic of Mary Lee coal was further emphasized in a crushing test, which indicated that crushing definitely reduced the quantity of material remaining in a given range of specific gravity. The table-products curve shows that up to 80 per cent recovery a satisfactory product can be obtained. With a feed coal of 19.0 per cent ash these tables are producing a washed coal of 10.75 per cent ash and a refuse containing 9.0 per cent float material at 1.40 specific gravity.

Influenced largely by the favorable results obtained from the Barney plant, in

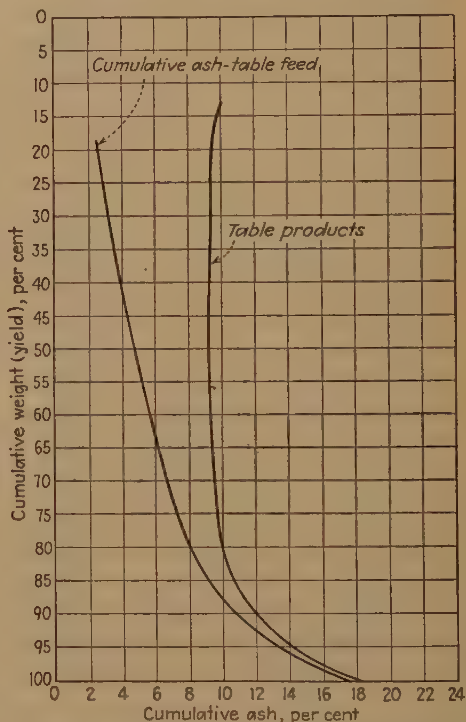

Fig. 2.—Typical zone-sample table test on diagonal-deck table as compared to float-and-sink study of feed.

1934 and 1935 the company installed eight rectangular-deck tables at its Colta mine and seven diagonal-deck and five rectangular-deck tables at the Praco mine. The primary purpose of this installation was to secure a low-ash coking coal for the by-product plant. As the coke plant has more exacting requirements as to low ash, uniformity in ash content, and absence of particles of noncoking nature than the average steam trade in Alabama, most of this paper will be devoted to the results obtained from these tables and the operating factors involved.

Like the Barney mine, the Praco and Colta mine operate on the Mary Lee

seam, but at those mines the coal is of a different character. The seam changes in analysis and physical characteristics from its western outcrop, where the Barney mine operates, to its eastern outcrop, where the Praco and Colta mines are situated. These last two mines are within 3 miles of each other. In the eastern section the coal is softer and lower in volatile matter, and consequently has better coking qualities. Also, it has somewhat less inherent ash. As the coal at Praco and Colta are similar in washing characteristics and the flow sheets of the two plants are practically identical, they will be described as one unit.

Prior to the installation of tables, the by-product plant has been furnished with a 1-in. to 0 coal washed on jigs and containing from 10.75 to 11.0 per cent ash. Since the tables have been in operation, the plant has been shipped an average of 1900 tons per day containing 9.50 per cent ash. This has been done with much less variation from minimum to maximum ash content than previously had been possible.

Table 2 gives float-and-sink data on Praco 3-in. to 0 and 3⁄16-in. to 0 size, respectively. Again will be noted the greater washability of the finer sizes. Fig. 2 shows a typical zone-sample table test on a diagonal-deck table at Praco as compared to a float-and-sink study of the feed. It might be well to point out that in these curves the float-and-sink data are on 3⁄16-in. to 0 size whereas the table products are from 5⁄16-in. to 0 feed. This would tend to make an efficiency calculated from these curves lower than normal. Table 3 gives an average analysis of the 5⁄16-in. to 0 table feed and washed coal at the Colta plant.

One of the earliest problems to develop was the necessity of producing a middling product and the utilization of this middling.

TABLE 1.—*Float-and-sink Data of Minus 3-inch Raw Coal, Mary Seam Lee at Barney Mine*

Description	Specific Gravity	Weight, Per Cent	Ash, Per Cent	Cumulative Weight, Per Cent	Cumulative Ash, Per Cent
Composite all sizes, minus 3-in. Weight per cent, 100.0.	Float on 1.28	4.5	2.7	4.5	2.7
	1.28–1.30	3.2	4.0	7.7	3.2
	1.30–1.38	59.9	11.2	67.6	10.3
	1.38–1.50	18.3	19.1	85.9	12.2
	1.50–1.70	4.9	32.6	90.8	13.3
	1.70–1.90	1.9	49.5	92.7	14.0
	Sink in 1.90	7.3	77.0	100.0	18.6

TABLE 2.—*Float-and-sink Data*

Specific Gravity	Weight, Per Cent	Ash, Per Cent	Cumulative Weight, Per Cent	Cumulative Ash, Per Cent
PRACO 3-IN. TO 0 RAW COAL				
Float 1.26	5.1	2.3	5.1	2.3
1.26–1.30	10.9	5.2	16.0	4.3
1.30–1.38	50.7	10.0	66.7	8.6
1.38–1.50	14.4	18.7	81.1	10.4
1.50–1.70	6.2	31.4	87.3	11.9
1.70–1.90	1.8	47.3	89.1	12.6
Sink 1.90	10.9	79.3	100.0	19.9
PRACO 3⁄16-IN. TO 0 RAW COAL				
Float 1.26	18.0	2.2	18.0	2.2
1.26–1.30	23.3	5.2	41.3	3.9
1.30–1.38	28.7	10.2	70.0	6.5
1.38–1.50	11.7	18.4	81.7	8.2
1.50–1.70	6.4	31.4	88.1	9.9
1.70–1.90	2.7	48.1	90.8	11.0
Sink 1.90	9.2	78.4	100.0	17.2

TABLE 3.—*Typical Colta Washed Products*

Product	Ash, Per Cent	Float 1.40		Sink 1.40	
		Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
Washed coal	9.56	92.6	8.0	7.4	27.7
Raw coal	22.12	74.7	8.1	25.3	62.1
Middling	18.25	68.8	8.9	31.2	35.7
Refuse		6.0		94.0	

Much experimental work has been done on this phase of table preparation both in our own plants and in cooperation with the Southern Experiment Station of the Bureau of Mines.

The bone and "bony coal" are less friable than the coal, therefore occur in relatively larger sizes. This condition, combined with the movement of extremely fine sizes of coal toward the refuse end owing to inherent sizing action of the table, produces a middling composed of fairly coarse high-gravity bone together with fine-size low-gravity coal. The middling cut is made several inches on both the washed coal and refuse side of table. This material constitutes 11.0 per cent of the table products and contains from 18.0 to 20.0 per cent ash.

The first step in the re-treatment of the middling was to wash this product on a separate table. Table 4 gives a typical zone test on the Praco middling. A washed coal, zones 1 to 7 inclusive, was produced at 15.89 per cent ash. These results proved very discouraging, as the washed product was not sufficiently free of bone to combine with the primary washed coal.

TABLE 4.—*Praco Middling, Table Products*

Zone	Weight	Weight, Per Cent	Ash, Per Cent	Cumulative Weight, Per Cent	Cumulative Ash, Per Cent
1	34.18	36.23	12.92	36.23	12.92
2	20.17	21.38	15.62	57.61	13.92
3	8.06	8.55	15.92	66.16	14.18
4	8.25	8.75	16.33	74.91	14.43
5	4.56	4.84	18.76	79.75	14.69
6	6.87	7.28	22.97	87.03	15.39
7	4.37	4.63	25.31	91.66	15.89
8	7.06	7.48	39.22	99.14	17.65
9	0.81	0.86	67.66	100.0	18.08

During 1937 the company, jointly with the U. S. Bureau of Mines, studied the possibility of classifying the middling to make it suitable as a table feed. It was apparent immediately that by reversing the size-specific-gravity relationship in the particles of bone and coal the classifier overflow could be efficiently tabled. Acting on the conclusions gathered from these experiments the company installed at the Colta plant a locally designed constriction-plate classifier and diagonal-deck re-wash table. The re-wash middling was passed directly into the primary washed coal from the eight rectangular-deck tables.

Table 5 shows an analysis of the products involved in the middling treatment.

TABLE 5.—*Analysis of Products in Middling Treatment*

Products	Ash Per Cent	Float 1.40		Sink 1.40	
		Wt. Per Cent	Ash, Per Cent	Wt. Per Cent	Ash, Per Cent
Classifier overflow...	16.48	74.3	8.66	25.7	33.33
Classifier spigot..... Product............	27.25	31.2	12.00	68.8	34.18
Washed middling....	13.18	76.9	8.30	25.1	26.87
Refuse from table...	31.98	25.0	8.65	75.0	39.66

By means of the classifier and re-wash table, 56.0 per cent of the middling was recovered as washed coal containing 13.0 to 13.5 per cent ash.

Although the middling was definitely improved by re-treatment, particles of non-coking bone in the by-product oven mix caused the coke to fracture excessively. Therefore, as a solution to this problem the by-product plant equipped its boilers with facilities for firing with pulverized coal, so that the table middling could be used to raise steam.

It has been the observation in this work that the limiting maximum size range of the table-feed coal is an important factor in efficient table operation. In normal practice screen sizes from 5/16 to 7/16 in. have been used and for test purposes ½ in., ¾ in. and 1 in. The ¼-in. to 0 size gave the most favorable results. However, because of the demand for increased quantity and to avoid additional crushing, a ⅜-in. to 0 size has proved most desirable. In all cases it was found that as the size of feed

coal was increased, the ash in the washed coal also increased. This increase was more pronounced above ½-in. size.

At Praco the elongated flat particles of bone or rash, as produced by slotted screen openings, were very difficult to separate on the table. Consequently part of this high-ash, noncoking material went into the washed coal. By changing to a square screen opening for sizing all table feed coal the oversize flaky bone was eliminated and a lower ash was obtained in the washed coal.

A uniform gradation from maximum to minimum size in the feed is desirable. An excess quantity of minus 20-mesh coal tends to increase the percentage of coal appearing as middlings and to increase the quantity of float coal in the refuse. An excess of near maximum-size particles tends to produce lack of uniformity in the table bed and causes the ash to increase in the first 2 ft. of the washed-coal discharge. For these same reasons segregation of feed coal either in storage bins or on conveyors should be avoided.

Because of fluctuations in the demand for table coal, additional raw coal at intervals must be crushed to supplement the normal 5⁄16-in. to 0 size run-of-mine coal. For this purpose, a 42 by 48-in. swing hammer pulverizer is used. To avoid an excessive quantity of fines in the table feed, it was found desirable to use extremely wide bar spacing in the hammer mill and a minimum of hammers. This crushing was necessarily in closed circuit. Another advantage of screening after crushing is that all particles larger than average maximum size are eliminated.

A uniform rate of feed to the tables is of vital importance. We have found that a 9-in. spiral conveyor operating at 14 to 16 r.p.m. constitutes a satisfactory table feeder, for a relatively dry raw coal; other methods of feeding would have to be employed with wet coal.

When washing Mary Lee raw coal on 14-ft. Plat-O tables and No. 7 Overstrom tables we have found that the practical range of feed tonnage is from 7 to 9 tons per hour. We have been unable to obtain less than 10.0 per cent ash when feeding over 9 tons per hour, and the best results have been obtained with a feed of only 8 tons. Since the last tables were installed in 1935 the manufacturers have brought out a new head motion for the diagonal-deck table, also a 17-ft. rectangular table with radical changes in deck design. This equipment was designed primarily to increase the capacity per table and this increase the equipment undoubtedly achieves, but we have not had sufficient experience yet with either of the changes to determine their advantages.

The volume of wash water required varies with the size of the feed and tonnage handled, and our requirements are 1.7 of water to 1 of feed. The water should be supplied at a uniform pressure, preferably from a constant-head reservoir. The water feed pipes should be of such diameter as to avoid high discharge pressure at the water boxes, and the valves should be maintained in good operating condition. Wash water at Colta and Praco consists of circulated water from the primary settling tank mixed in the table header line with about 15 per cent of fresh water. This circulated water contains an average of 10 per cent solids at 18.0 per cent ash. This is not particularly detrimental to table operation, but the solids should not exceed 12 per cent.

Unless tables are properly operated and maintained, they can become more liability than asset. An experienced attendant is a necessity. He should be able by glancing at the table bed to decide whether the table is operating at normal efficiency. Instant adjustment of wash water is required when minor variations occur in the feed. An evenly distributed mobile bed is essential at all times. Bunching of coarse coal or "rivers" running across the table should be avoided.

Rubber covers and rubber riffles have been found economical. The riffles on the rectangular-deck tables are ⅝ to ¾ in. high and on the diagonal-deck tables ⅞ to 1 in. The diagonal-deck tables also have "slate" riffles paralleling the back of the table. As washability and the most profitable separation vary widely in the different types of coal, details of final setting and adjustment of tables would be too voluminous to include in this paper. Extensive experiments have been made for our work over a wide range of speeds, lengths of stroke, heights of riffles, and elevations, cross pitches and other operating variables.

Proper maintenance is conducive to a successful operation of a table plant. Worn riffles should be replaced when their height has been reduced to such an extent that washing efficiency is impaired. Holes or leaks in deck covering should have immediate attention. Leaks around the feed corner and back board should not be allowed. Feed boxes on the diagonal-deck tables and the primary wash box on the rectangular-deck table should be kept in good condition. The latter items are important because they affect the immediate distribution of the feed coal as it arrives at the table.

One operator can supervise 12 tables and one man-shift per six tables per week will take care of all minor repairs. With normal care decks and drive mechanism will give 8 to 10 years service before major repairs are required.

With 8 tables at Colta and 12 tables at Praco, the by-product plant has been furnished with a coal more suitable than ever before for the manufacture of metallurgical coke. Though our experience with tables has not led to the belief that they are the perfect or ultimate method of preparing the finer sizes in all seams, the results have been satisfactory in our operation and much has been accomplished that could not otherwise have been attained.

DISCUSSION

(R. Dawson Hall presiding)

C. J. Potter,* Indiana, Pa.—Do you obtain effective cleaning on the sizes below 28 mesh in wet washing of the ⅜-in. to 0? What is the normal practice to recover and dewater the ⅜-in. to 0?

J. Griffen,† Pittsburgh, Pa.—Messrs. Hager and Haskell have given very full and valuable data on the practices they have developed, and most of the cautions they have mentioned are general in their application.

Their study of the re-treatment of primary table-middlings product is interesting, particularly the use of hydraulic classification and the tabling of the classifier overflow product. It is unfortunate that the water needed for classification is not given. It is probable that equally good results could be obtained by screening the middlings at a mesh that would remove the coarse bone and the screen undersize tabled. Water consumption would be materially reduced and it is possible that the bone, which interfered with coke structure, would be more completely removed. The Pittsburgh Coal Co. is screening ⅜-in. to 0 middlings from a Rheolaveur unit at 3⁄16 in. and tabling the 3⁄16-in. to 0 product, with quite satisfactory results.[1]

The test work done jointly with the U.S. Bureau of Mines mentioned by the authors is undoubtedly that covered by *Report of Investigations No.* 3448. That publication gives screen tests on the primary table middlings, which indicate that screening at 8 or 10 mesh would be a promising line of attack.

H. J. Hager (author's reply).—Answering Mr. Potter's questions: We do not obtain efficient cleaning on minus 28-mesh size; however, we do feel that we get satisfactory results as compared with cleaning this size by any other method.

I do not have actual screen and ash analysis on the 28-mesh to 0 size, but Table 6 gives analysis to 1⁄16-in. to 0.

* Rochester and Pittsburgh Coal Co.
† McNally Pittsburg Manufacturing Corporation.
[1] J. T. Crawford, C. P. Proctor and J. A. Younkins: Launder and Table Washing of Fine Coal. *Trans.* A.I.M.E. (1940) **139**, 285.

TABLE 6.—*Screen Analysis on 3/8-inch to 0 Washed Table Coal, Colta Mine*

Size, In.	Weight, Per Cent	Ash, Per Cent
3/8 to 1/4	5.50	11.58
1/4 to 1/8	27.00	9.97
1/8 to 1/16	31.75	8.45
Minus 1/16	35.75	9.60

Cumulative ash per cent at 80 per cent recovery from actual zone-sample table test, Praco mine, is as follows:

Size, In.	Ash, Per Cent
Plus 1/4	12.26
1/4 to 1/8	11.20
1/8 to 1/16	9.10
1/16 to 0	9.40

The 1/16-in. to 0 size being higher in ash than the 1/8-in. to 1/16-in. definitely indicates that at some point below 1/16-in. the cleaning efficiency is dropping.

Our normal practice in dewatering and recovering the 3/8-in. to 0 is as follows: The washed coal is flumed by 1/2-round iron trough to a 12 by 40-ft. settling tank; thence by perforated bucket elevator to a chute containing a section of wedge-wire dewatering screen. The opening in the screen is 1/4 mm. The screen chute discharges into a 50-ton storage bin. The pitch of the screen is adjustable and so set that one elevator bucket of coal stays on the screen until pushed off by the next bucket. Water through the screen is returned to the settling tank.

The settling tank is equipped with sludge-recovery drag conveyor. A circulating pump takes overflow water from the tank directly to the table feed-water header line.

Under this system the 3/8-in. to 0 coal is loaded into railroad cars at about 18 per cent moisture and on 24-hr. transit is received at the by-product at 8.0 per cent moisture.

Progress in Air Cleaning of Coal

By David R. Mitchell,* Member A.I.M.E.

(New York Meeting, February 1942)

This paper is limited primarily to a description of dry coal-cleaning processes in which air currents are used to effectuate a separation between coal and refuse. Processes depending mainly on differences in the coefficient of friction, resilience and shape are not described unless the use of air is necessary for the proper functioning of the process.

The air cleaning of coal is of primary importance to the coal industry of the United States. From figures presented in the 1941 edition *Mechannual*,[1] it is estimated that at least 12 million tons of bituminous coal was cleaned in the United States during 1940 by machines using air as the separating medium.

One of the current problems confronting the coal-preparation engineer is the dry cleaning of small sizes and slacks. Progress in machine development for the cleaning of small sizes has been slow in recent years in the United States of America; it has been much more rapid in Britain and on the European continent. This paper does not attempt an exhaustive survey of all these developments, and only machines and processes that have found considerable commercial application are cited. Further, no critical analysis has been made of the theory involved in the separation of coal from refuse by processes using air as the separating medium other than that necessary for an understanding of particular machine designs and applications.

Historical Summary

The possibility of using air for the separation of coal from refuse was recognized early by mining men and many attempts were made to develop commercial machines. The ingenuity of the early experimenters is to be marveled at in view of their lack of knowledge of the theoretical principles involved. As in other arts, practice has largely preceded theory. Yet, science and practice play a game of leap frog, now one ahead, then the other; the careful experimenter governs his efforts along practical lines by adapting his experiments to the limits prescribed by scientific laws. By first studying developments and the theory involved, it is not only possible to adapt processes to new uses but also to reduce materially operating cost and to build larger units without costly failures.

Although many inventors have applied for patents for separating minerals by the use of air currents, the literature in regard to coal is meager. Berrisford[2] describes briefly some of the early attempts to clean coal without the use of water. Several of the examples cited by him are reproduced herein.

Fig. 1 shows a machine used for grinding and; izing, in which rollers running in a pan with a perforated bottom and fan opening on the side crushed the coal, which was then carried by the air currents to a settling chamber. The heaviest particles dropped out first and the light ones farthest away, as shown.

In the year 1864, a machine was made that could be used as a sizer or to separate dirt from coal with a sized feed (Fig. 2).

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[1] References are at the end of the paper.

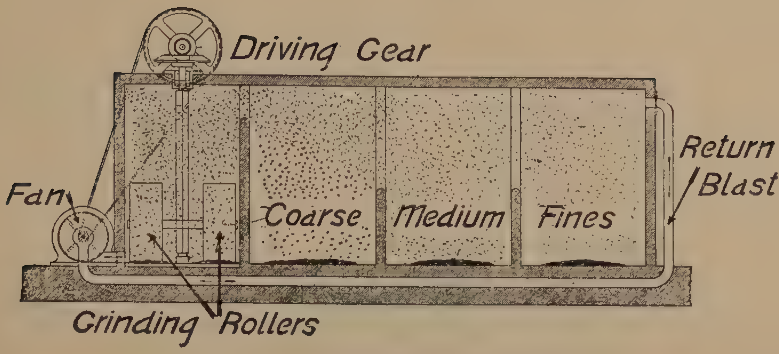

FIG. 1.—COAL-CRUSHING MACHINE (1859).

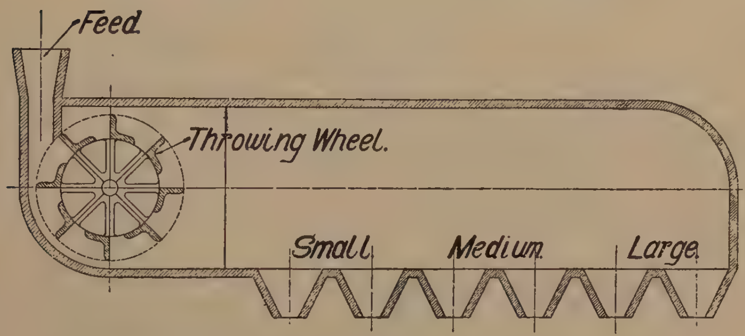

FIG. 2.—MACHINE FOR SEPARATION OF COAL FROM DIRT (1864).

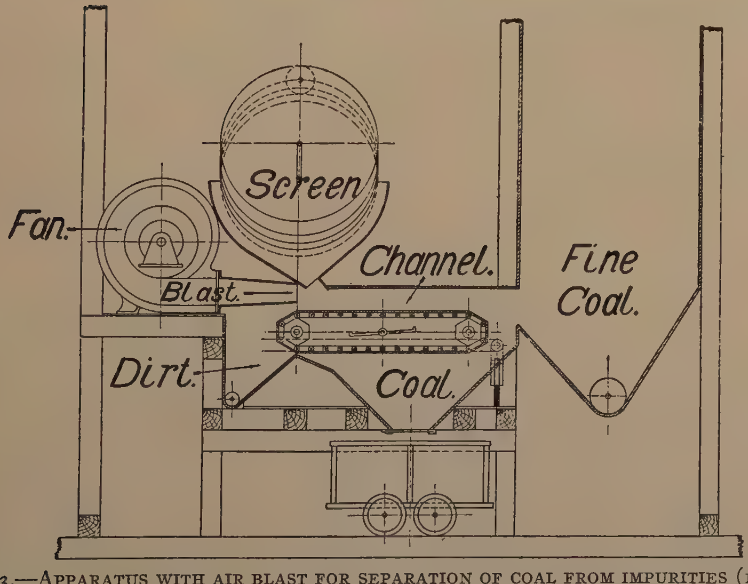

FIG. 3.—APPARATUS WITH AIR BLAST FOR SEPARATION OF COAL FROM IMPURITIES (1878).

After being thrown out, the products settled according to size and specific gravity.

Fig. 3 shows a machine that used an air blast to move coal along a plate belt moving blast of air was used to assist the separation —one of the few examples in which downward currents of air are used to assist separation; in fact, downward currents are rarities in wet-type separators.

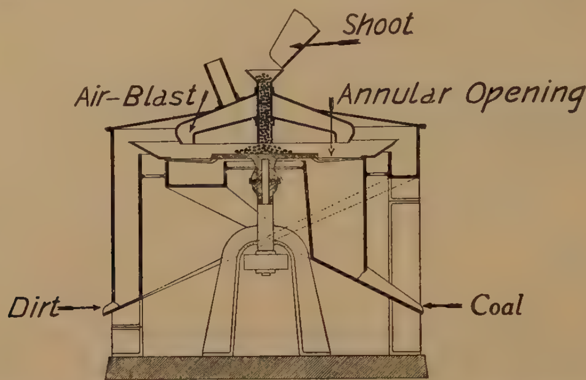

Fig. 4.—Machine with rotating disk for separation of coal from dirt (1909).

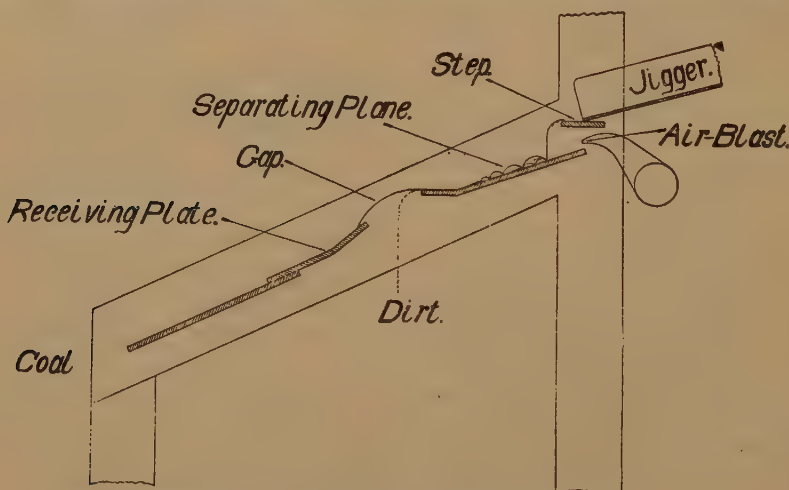

Fig. 5.—Berrisford machine for separation of coal from dirt.

in the opposite direction to the blast of air. Refuse clung to the belt and was carried to the end nearest the blast. In this machine the coal was closely sized, each size dropping into a channel along which the air from the fan passed. The floor of the channel consisted of this moving plate belt.

Centrifugal types of machines have been tried at various times. Fig. 4 shows an interesting example of a type in which a

Coal cleaners based on the difference in the coefficient of friction of coal and refuse were made as early as 1868, important developments being the introduction of the spiral (Pardee) separator in 1898 and the Langerfield in 1903. Also in 1903, it was generally recognized that coal and associated impurities differ in shape, and separators based on this fact began to appear. In 1925 the Messrs. Berrisford

introduced to the trade an improved machine for the separation of refuse from sized coal, employing not only differences in the coefficient of friction but also differences in resilience and specific gravity. This machine (Fig. 5) employs an air blast to keep the plates clean. It is made with two or three stages and has been found to be remarkably efficient in cleaning the nut sizes of some coals. Approximately 200 of these machines are reported to be in operation.

A large group of concentrating devices using air as the separating medium have been devised for the treatment of ores. Many of them—jigs, launders and tables—have been modified in design and applied to the cleaning of coal.

Henry M. Sutton, Walter L. Steele and E. G. Steele, organizers of the well-known firm of Sutton, Steele, and Steele, became interested in air cleaning in 1902. They pioneered developments in the United States and invented the first commercially

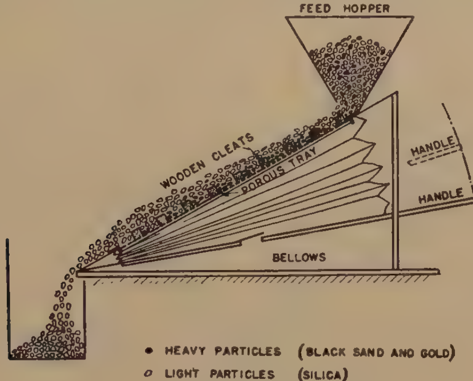

FIG. 6.—THE ARIZONA WASHER.

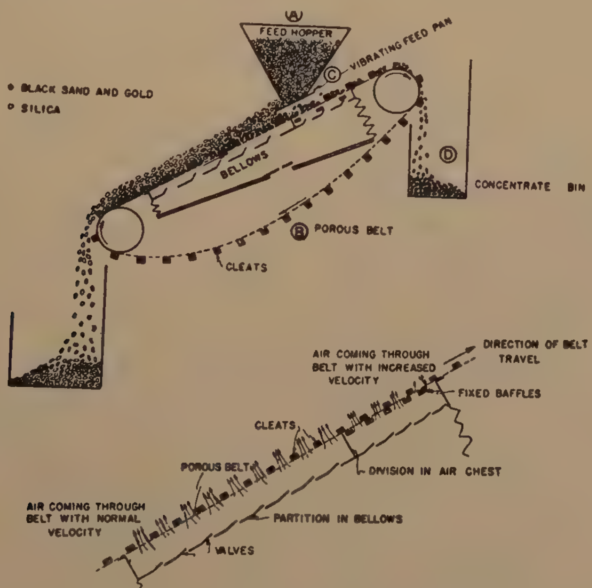

FIG. 7.—PNEUMATIC JIG FOR DRY PLACER.

successful air table for cleaning coal. Their first work was on ores; they mechanized the Arizona dry washer (Fig. 6), which was used on placer ores by adding a moving porous belt with transverse cleats (Fig. 7). R. R. Slaymaker[3] describes the various developments pioneered by the Sutton, Steele, and Steele organization.

Applying classification theories and calculations to the separation of coal from slate, it is found that a size ratio of 2 must be adhered to under free-settling conditions to obtain a separation. By developing machines to operate under hindered-settling conditions, by developing bed-type separation equivalent to the use of a dense medium, or by utilizing other physical properties of coal and refuse, it is possible to increase greatly this ratio. The air concentrator operates in an atmosphere of the same fluid that is used for separation, hence there is no difference in viscous resistance to motion or in supporting effect on particles when they pass into or out of the medium, as in water types of concentrators.

Machines using air currents as the separating medium for the cleaning of coal may be roughly classified into four general groups: (1) jigs—stationary devices with pulsating air currents; (2) shaking tables—continuous upward air currents; (3) launders—continuous or pulsating air currents; (4) dense-medium processes.

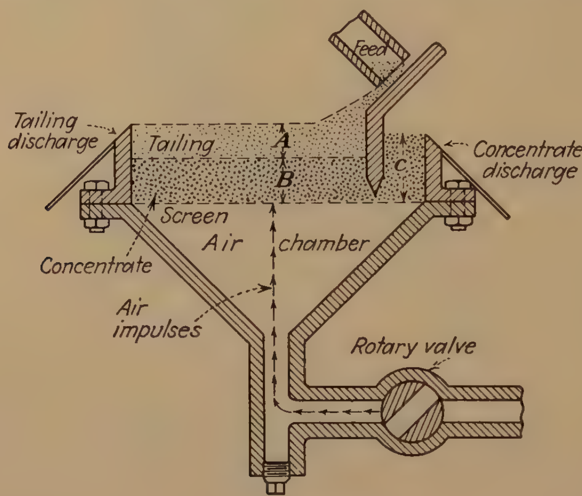

FIG. 8.—PLUMB JIG.

JIGS

Plumb Jig

One of the earliest jigs to be tried experimentally for cleaning coal was the Plumb, which had found some use in the concentration of ores. A cross section of this machine is shown in Fig. 8. The separating compartment is 3 in. wide and 24 to 36 in. long. With a closely sized feed good separation can be obtained, but because of low capacity and difficulties of screening small sizes of coal, with consequent high cost of operation, there is no record of this machine being used commercially in coal-cleaning plants.

The Kirkup Separator

The Kirkup separator, developed in England, and applied to the cleaning of coal at Consett, Durham County, is a stationary deep-bed separator of much larger size and capacity than the early air jigs used for

concentrating ores. One of the first types is shown in cross section in Fig. 9. The separating trough is 14 ft. long and 2 ft. wide, with sides 12 in. high set at an angle of

TABLE 1.—*Results of Cleaning on New Kirkup Separator*[4]

Coal	Size, Mm.	Ash, Per Cent Raw Coal	Ash, Per Cent Clean Coal	Middlings	Dirt	Dust
Westphalian..	0.5– 3	29.0	16.9	54.8	78.0	39.2
	3 –10	17.7	7.4	44.1	80.1	22.6
Upper Silesian	10 –40	18.9	9.2	47.5		
Ruhr..........	0.5– 3	21.7	7.4	42.7	78.7	15.3

from 12° to 15° from the horizontal. A bed of coal 6 to 8 in. deep is maintained in the machine, and is stratified by the pulsating air current, which fluidizes the bed sufficiently to make the coal flow down the inclined trough and to loosen the bed so that individual particles will arrange themselves according to differences in specific gravity. Rated capacity on 1½-in. screenings is 25 tons of feed per hour.

The original Kirkup separator was developed by Dickinson and Kirkup during the period 1924 to 1928. Subsequently work was carried out in Germany in collaboration with a German firm and several plants were built on the continent of Europe.

A new design (Fig. 10) was introduced in England at Consett in 1931. Instead of treating minus 1½-in. to 0 coal in one stage,

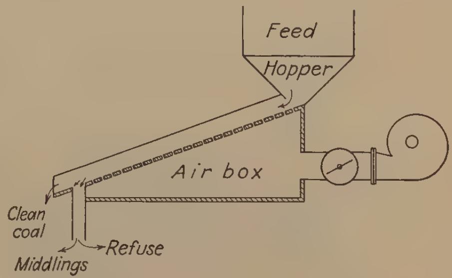

FIG. 9.—KIRKUP SEPARATOR.

the coal is sized at ⅜ in. The 1½ to ⅜-in. size is treated on one separator and the minus ⅜-in. on another separator. The separator has two stages. The deck is convex and only about 4 ft. long for each stage. Machines can be built with three or more stages. This new separator is built either 3 ft. or 3 ft. 8 in. wide. Capacities as high as 57 tons per hour on the 1½ to ⅜-in. size and 27 tons per hour on the minus ⅜-in. size have been achieved.

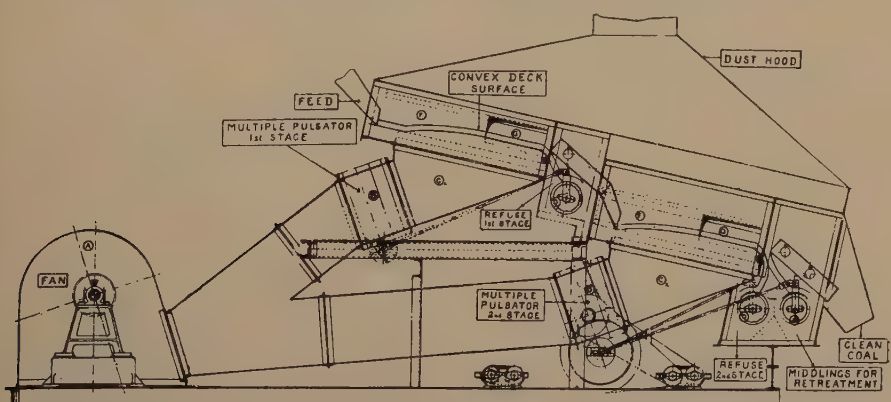

FIG. 10.—DIAGRAMMATIC REPRESENTATION OF NEW KIRKUP PROCESS.

Mott[4] gives the examples of cleaning results shown in Table 1.

Results given in *Colliery Engineering*[5] at an installation in Durham County, England, cleaning 1½-in. screenings in two sizes on two separators are as follows: raw

coal to plant, 9.5 per cent ash, 7 to 8 per cent water; clean coal, average 6 per cent ash; float at 1.50 sp. gr. averages approximately 97 per cent.

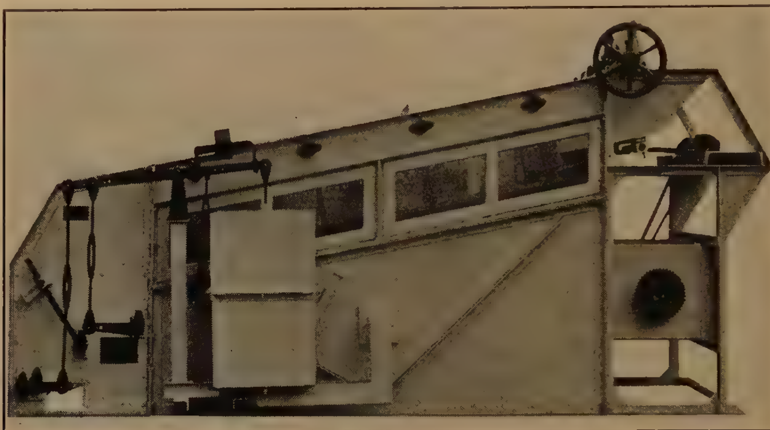

FIG. 11.—STUMP AIR-FLOW SEPARATOR.

Capital costs of a Kirkup plant are estimated at $300 to $400 per ton of hourly capacity. Power requirements are approximately 0.5 hp. per ton, excluding dust removal and collection.

Stump Air-flow Coal Cleaner

The air-flow cleaner was developed by Earl Stump in 1932, at the testing plant of the Roberts and Schaefer Co., Harvey, Ill. The first commercial installation was made that same year for the Barnes Coal Co., in central Pennsylvania. This machine is well known to American coal-preparation engineers and has been widely used. During the first six years it was on the market, 98 machines were installed. Fig. 11 shows an air-flow cleaner equipped with a dash pot. Improvements in design have been rapid. The first machines were 18 in. and 24 in. wide. The width has been increased nearly every year. All installed in 1938 were 6 ft. wide, except one 4-ft. machine.

This machine consists of a stationary, perforated deck, 8 ft. long, with a slope of 2¼ in. per foot, set over a bed of marbles to get even distribution of the pulsating air through the deck. The over-all length of the 1938 machine is 12 ft. The thickness of the bed of marbles varies from 8 in. at the feed end to 3 in. at the discharge end. The sides of the machines above the deck are made of glass plates and a dust hood completely encloses the unit. The plenum chamber below the deck and bed of marbles is tightly enclosed and connects through the flutter valve compartment to a common plenum chamber with other units.

A reciprocating plate (known as the zoning plate) is suspended above the bed plate. Originally a perforated plate, it is now constructed of square-mesh wire screen cloth. It is suspended from steel rods to within 2 to 3 in. of the deck and attached to the feeder plate, which reciprocates with a 1-in. stroke. The purpose of this zoning plate is to prevent boiling of the bed. It also assists the movement of coal down the deck and tends to prevent entrapment of particles.

The bed of refuse concentrated on the deck of the machine increases in thickness toward the lower discharge end, and for this reason the marble pack decreases in thickness from the feed to the discharge end, so that the resistance of the bed and pack from feed to discharge will be approximately the same. The original machine has an automatic refuse-discharge gate operated

through a dash pot, which in turn is operated by the air pressure of the cleaner. The dash pot consisted of a float in a bath of water or oil and a pipe extending from under the float to the plenum space under the marble pack. The float was connected by levers to the discharge gates. A build-up of pressure in consequence of an increase in thickness of the bed of refuse was supposed to increase the pressure in the plenum, which would cause the float to rise, and thus, through the system of levers, open the refuse gates wider. Since the discharge of products was most irregular, this method has been replaced by a rotary valve with a variable-speed drive, and has proved much more satisfactory than the dash-pot method.

TABLE 2.—*Sizing and Gravity Separation Characteristics of Air-float Products, Coal C, Minus 5/16 Inch*

(AFTER DAVIS AND HANSON)

Material	Size, Inches and Tyler Mesh	120 Tons per Hour—Input			
		Percentage of Material[a]	Percentage of Size	1.60 Float, Per Cent Weight	1.60 Sink, Per Cent Weight
Raw coal (feed)	−5/16″	100.0	100.0		
	5/16″ to 48 m.		90.0	92.3	7.7
	5/16″ to 14 m.		69.5	92.8	7.2
	14 m. to 48 m.		20.5	90.6	9.4
	−48 m.		10.0		
Primary clean coal, two 6-ft. units.	−5/16″	82.5	100.0		
	5/16″ to 48 m.		87.5	96.3	3.7
	5/16″ to 14 m.		61.0	98.2	1.8
	14 m. to 48 m.		26.5	91.9	8.1
	−48 m.		12.5		
Primary refuse and middlings, two units.	−5/16″	15.5	100.0		
	5/16″ to 48 m.		97.0	72.2	27.8
	5/16″ to 14 m.		95.5	73.3	26.7
	14 m. to 48 m.		1.5	16.7	83.3
	−48 m.		3.0		
Secondary feed (includes secondary middlings recirculation).	−5/16″	23.5	100.0		
	5/16″ to 48 m.		98.0	78.1	21.9
	5/16″ to 14 m.		96.5	78.4	21.6
	14 m. to 48 m.		1.5	50.0	50.0
	−48 m.		2.0		
Secondary clean coal, one 4-ft. unit.	−5/16″	11.0	100.0		
	5/16″ to 48 m.		98.5	95.2	4.8
	5/16″ to 14 m.		97.0	95.9	4.1
	14 m. to 48 m.		1.5	33.4	66.6
	−48 m.		1.5		
Secondary middlings.		8.0	No sample		
Final refuse (secondary unit).	−5/16″	4.5	100.0		
	5/16″ to 48 m.		99.0	23.4	76.6
	5/16″ to 14 m.		97.5	23.6	76.4
	14 m. to 48 m.		1.5	9.1	90.9
	−48 m.		1.0		
Cyclone product		2.0			
Final clean coal[b]	−5/16″	95.5	100.0		
	5/16″ to 48 m.		89.0	96.1	3.9
	5/16″ to 14 m.		70.0	97.5	2.5
	14 m. to 48 m.		19.0	91.0	9.0
	−48 m.		11.0		

[a] The percentage weight of material given is the approximate percentage of raw coal fed to the plant.
[b] Includes primary clean coal, secondary clean coal and cyclone products.

The clean coal is separated from the refuse or middlings by a horizontal cutting plate. In the original machine damp material tended to stick to this plate. The use of a stainless-steel plate with edges cut at 60° has removed this troublesome condition.

In the 1938 model of the Stump air-flow cleaner the dash pot is replaced by a rotary valve. Davis and Hanson[6] presented a critical analysis of the changes incorporated in this new model. They also report detailed tests on the operation of these units on three coals from the Pittsburgh bed in western Pennsylvania, but from different mines. Table 2 gives the results they obtained on coal C. The coal at this plant was first cleaned on three units, each 6 ft. wide, making a clean coal and a middling-refuse product. The middling-refuse product from these primary units was further cleaned in a secondary unit, 4 ft. wide, with recirculation of middlings through the unit. Each unit was preceded by a 25-ton surge bin.

These machines are usually used for minus 5/16-in. coal. The best size range is from 2 to 2½ to 1; the recommended sizes for cleaning being 1¼ to ¾, ¾ to 5/16, and 5/16 to 0 inches.

It is to be noted from Table 2 that there is very little cleaning of the minus 14-mesh material. A lowering of the top size would help this, undoubtedly, although it is not likely that any reduction of impurities could be effected for sizes smaller than 48 mesh. Minus 48-mesh material is largely removed as a dust product. The capacity of a unit varies from ½ to ¾ ton per hour per inch of width.

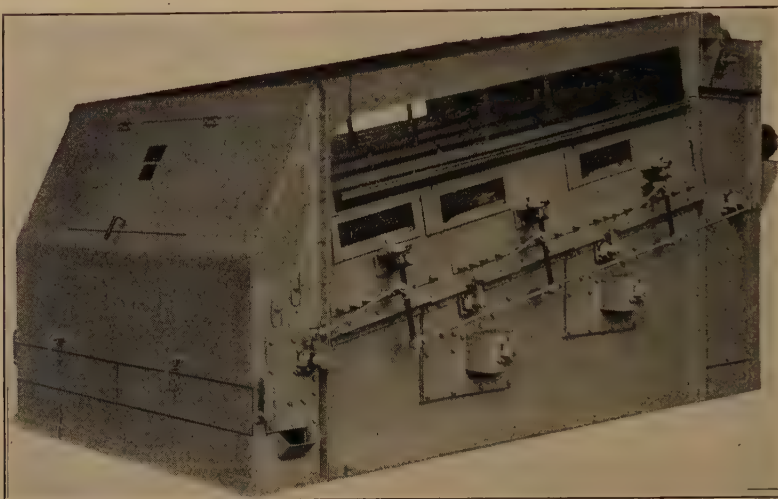

FIG. 12.—STUMP AIR-FLOW CLEANER.

In recent models the rotary valve has been replaced by a reciprocating refuse gate. One of these machines is shown in Fig. 12.

SHAKING TABLES

Concentrating tables using air as the separating medium were developed first for ores, mainly for the concentration of gold, in arid regions where water was not available for the operation of wet processes. Tables of various designs were developed by Card, Sutton, Bonson, Stebbins and others during the period 1890 to 1910, and received some commercial success in the West.

The first commercial installation for cleaning coal was of the type CJ, Sutton, Steele and Steele table, at a mine of the McAlister-Edwards Coal Co. near Pittsburg, Okla., in 1919. This table was designed for the cleaning of ores, grains, and seeds. Following this initial installation the American Coal Cleaning Corporation was formed to develop this and subsequent

models. Development was rapid and various types of American separators evolved based on the original Sutton, Steele and Steele patents and designs. The SJ model

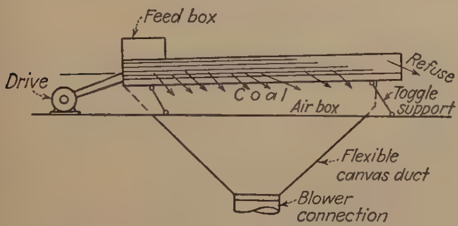

FIG. 13.—ELEMENT OF THE AIR TABLE.

was introduced in 1922, the Y in 1925, and the R or rectangular deck (similar to the Vee table developed by the Birtley company in England under American Coal Cleaning Corporation license), and the Twin-Dex in 1939, marketed by the Link-Belt Co. and the Fairmont Machinery Co. through a license agreement with the Peele-Davis and American Coal Cleaning Corporation. The Peele-Davis table was introduced in 1924, the Arms table in 1925, and the Heyl and Patterson in 1927. Developments in table-type separators have lagged in the United States of America during recent years but have been pronounced in Europe, where a number of different types have been introduced. A brief summation of the principal features and operating results of the better known tables follows.

Elements of the Air Table

Mechanical features of an air table consist of a pervious deck, transversely inclined, set horizontally or with a small upward inclination longitudinally, supported on rollers or toggles, or suspended by means of flexible hangers, so that it may be shaken back and forth in a substantially horizontal direction. Riffles extend across the deck in a longitudinal direction and the deck is connected to an air chest below, so that air may be blown up through the deck. These elementary features, more or less common to all designs, are shown in Fig. 13. The coal to be treated spreads out over the table surface, where, under the combined action of upward-flowing air currents,

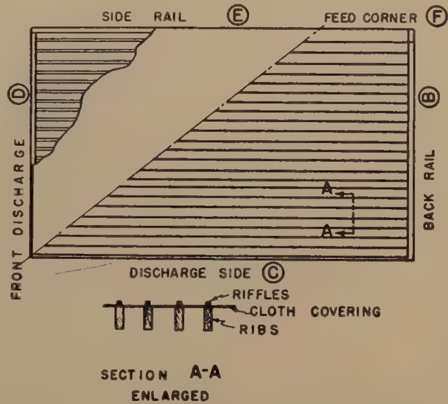

FIG. 14.—DIAGRAM OF FIRST PNEUMATIC DECK.

motion of the table, and frictional resistance of the table deck, the light coal particles stratify above the refuse particles and flow down over the side of the table, while the heavy refuse is trapped in the riffles and moves to the end of the table.

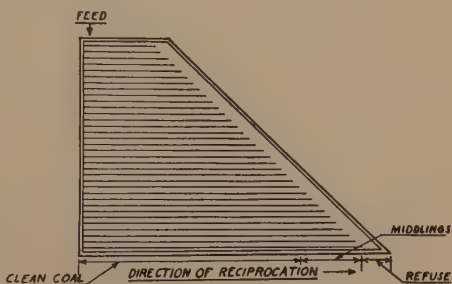

FIG. 15.—CJ TABLE.

The deck plans of the early air tables resembled those of wet ore-dressing tables, many being rectangular in shape. Fig. 14 is the deck plan of the first Sutton, Steele and Steele table, built in 1903. It was early noticed that the air table acted as both a sizer and a cleaner. Refuse arriving at the end of the riffles carried considerable large coal. It was found that a banking bar placed at about the position of the dotted

line in Fig. 14 caused the banked up refuse to squeeze out the entrapped coal particles, which then rolled back on to the table to the clean-coal or middling portion. Further development and research has led to the designing of decks without the unused space in the corners. Further improvements in banking bars, air controls, riffle designs, and drive features have made it possible to increase greatly the size range that tables will handle.

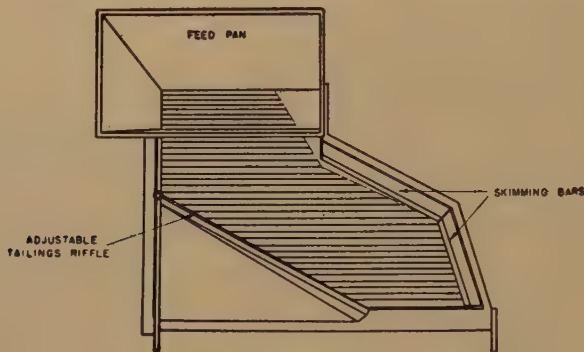

FIG. 16.—THE SJ 60 TO 84-INCH COAL DECK.

The CJ Table (1916).—The CJ table was a wide, short table, giving a long path of travel for the coal and a short distance for the refuse (Fig. 15). A diagonal banking board was used. Rejection of coal particles from the refuse zone was also assisted by a current of air directed transversely back over the deck by a curved deflecting plate at top of the banking bar.

The SJ Table (1922).—The SJ table was the first standard model designed specifically for the cleaning of coal (see Fig. 16 for deck plan). The principal improvements incorporated in the design of the SJ tables were: (1) changing the shape of the deck to conform to the actual working area of the table, (2) enlarging the deck to obtain greatly increased capacity and increasing riffle height, (3) improving air control and drive arrangements.

Riffles vary in height from 1¼ to 2 in. at the feed end to 3/32 in. at the refuse end, terminating along the refuse end on a line approximately parallel to the banking bar, thus leaving a narrow unriffled zone. Riffle spacing is usually 1 in. for small coal and up to 6 in. for nut sizing. Riffles were sometimes arranged in pools, so that the coal cascaded over the stepped riffle tops, causing flat pieces to turn on edge and thus facilitating the sinking of flat refuse to the deck surface with the rest of the refuse. The standard SJ, 60-84 machine, was approximately 60 in. wide and 84 in. long, with a rated capacity of approximately 15 tons per hour for ⅛ to 1/16-in. coal, and approximately 60 tons per hour for 2 to 4-in. coal.

The Y Table (1925).—The Y table, introduced in 1925, incorporated three new distinct features of design: (1) the deck is designed in the form of a letter Y, with the feed end at the base of the Y, which is approximately the shape obtained if two SJ tables are joined together; (2) tapered riffles are laid diagonal to the flow of coal instead of normal to the flow as in earlier models; (3) a gradation of the air blast was made.

In addition to the standard Y design shown in Fig. 17, a half-Y model was made for handling smaller tonnages and a Y-A model was designed with special provision for independent adjustment of the two halves, so that different sizes might be treated on the two halves.

The Vee Table (1928).—Cooperating with the American Coal Cleaning Corpora-

tion, the Birtley Iron Co. of England introduced into England the SJ (1925) and the Y (1927) separators, followed in 1928 by table, the deck plan of which is shown in Fig. 18.

Features incorporated in the Vee table

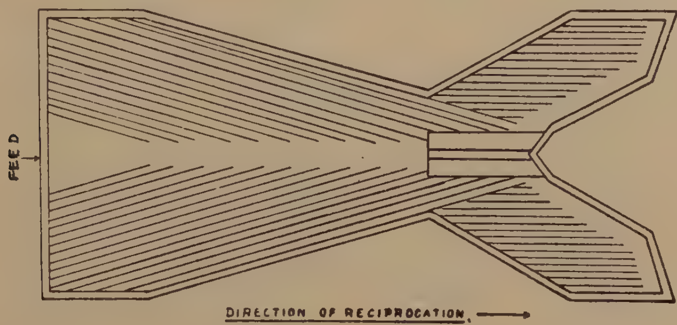

FIG. 17.—Y TABLE.

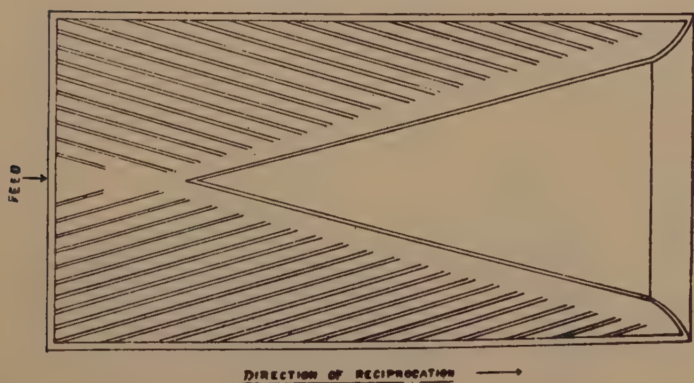

FIG. 18.—VEE TABLE.

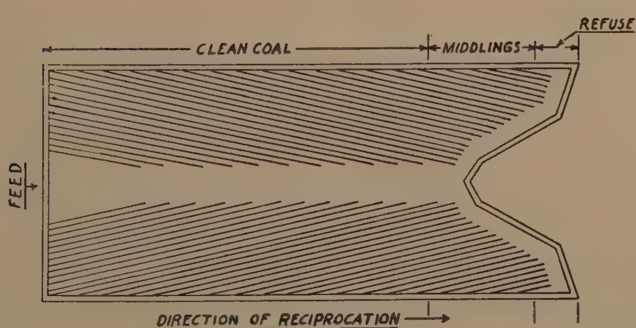

FIG. 19.—AMERICAN TYPE R.

the introduction of the Vee table, developed by the research staff of the Birtley company. Mott[4] describes in detail the development and operation of the Vee are: (1) simplification of the shape of the deck, (2) narrowing the width of the arms so that the coal is squeezed outward to spill over the long sides of the V, (3) a carefully

designed banking bar curved to get the best results in freeing refuse of entrapped coal, (4) the rectangular outline of the complete table, which simplifies support and drive features.

The effectiveness of this design is shown in the increased capacity per square foot of deck area and the ability to clean in the size ratio of 4:1 as compared with the 2:1 ratio needed for the SJ table. All Y tables installed in England have been replaced by the Vee type, with increases in capacity up to 30 per cent. Appleyard[7] reported in September 1930 that there were 70 or more plants in operation containing 221 of these machines.

Mott[4] says: "It is usually possible to clean the 2-in. to 0 raw coal so as to leave an average of 3 per cent of sinks at 1.5 S. G. in the clean coal, and to guarantee that the total clean coal is not more than 2.0 per cent higher than the fixed ash at 1.5 S. G., whilst leaving not more than 2.0 per cent of coal (floats 1.5 S. G.) in the refuse."

In treating minus ⅛-in. coal, cleaning may be effective to $\frac{1}{32}$ in. and with favorable coal to $\frac{1}{64}$ in.; in the latter case, a size ratio of 1:8 is in use.

American Type R (1930).—The American Type R table, shown in Fig. 19, is similar to the Vee table developed in England by the Birtley company. Three sizes are made: 6 by 12 ft., 6 by 14½ ft. and 8 by 17½ ft. Performance and operating characteristics are similar to those of the Vee table.

Twin-Dex (1939).—It is apparent from Fig. 20 that the manufacturers have returned to a deck shape similar to the original SJ table. Two modified rectangular decks with increased width of riffle area are mounted as one machine, the two decks occupying about the same space that would be required by one full rectangular deck. Each deck is individually adjustable as to transverse or longitudinal slope and toggle angularity, and has its own separate feed box, supply fan and air chest. The decks are arranged with their feed ends opposite one another and their discharge lips parallel, and are supported on toggles fitted with rubber bushings instead of the customary metal ones.

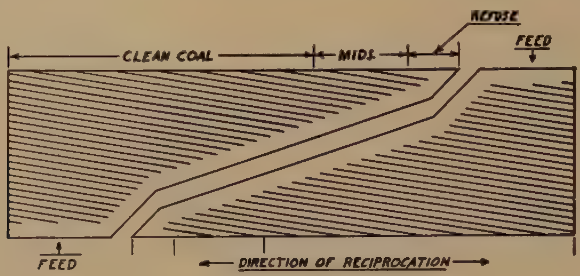

FIG. 20.—TWIN-DEX (1939).

These features provide maximum accessibility for machinery maintenance and product inspection, and with the decks counterbalancing one another vibration to the supporting structure is minimized, since the decks oscillate 180° out of phase. Each deck is entirely independent of the other except in the matter of speed.

The Twin-Dex separator has increased capacity and cleaning efficiency as compared with older models and has been well received.

The Arms Table (1923).—The Arms table was installed by the Roberts and Schaefer Co. at Covel, W. Va., in 1923. The deck plan is shown in Fig. 21. It more nearly resembles wet concentrating tables than any of the other tables described herein. Although the plan of the deck is rectangu-

lar, the area through which air is blown is nearly triangular, conforming in shape to the natural spread of the coal. This table is no longer on the market. Its chief features were the use of a high-resistance deck to get improved air distribution. The table for cleaning minus ¼-in. or ⅛-in. coal had a deck covered with a heavy close-woven woolen cloth, which not only promoted uniform distribution of air but also assisted in trapping refuse, much as blankets are used to recover gold at certain mills.

The Heyl-Patterson Table (1927).—The Heyl-Patterson table was first installed at the Orenda mine, of the Davis Coal and Coke Co., Boswell, Pa. The deck is nearly triangular in shape, narrowing toward the discharge end and leaving a narrow unriffled space between the riffle ends and the stepped, diagonal end wall. This feature is similar to the early Stebbins ore table. making two half tables, one of which may be used as a primary table and one as a re-treatment table. Run-of-mine coal up to 8 in. in size has been treated on one table.

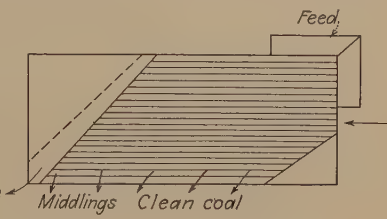

FIG. 21.—THE ARMS TABLE.

The riffles are laid diagonally to the direction of motion, with a slight traverse slope to the center of the table, making a valley. The air compartment is zoned; hence, by taking advantage of the natural sizing action of an air table, it is possible to clean coal of a wide size range. The refuse is carried out by the riffles to the side and the clean coal to the end. Fig. 22 shows the arrangement when two half tables are used separately, one as a primary cleaning unit and one as a re-cleaner.

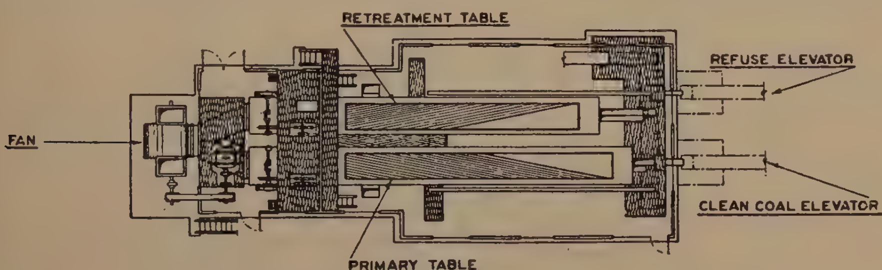

FIG. 22.—PLAN OF LAYOUT OF PEALE-DAVIS PLANT.

Performance and operating characteristics are similar to those of other tables of this general class. This table slopes downward from the feed to the refuse discharge end.

The Peale-Davis Table (1924).—The Peale-Davis table was developed at the mines of the Peale, Peacock and Kerr Coal Co. in central Pennsylvania and is similar in general operating characteristics to the tables previously described. It is radically different in that it is much larger than any other air table so far developed. It is approximately 14 ft. wide and 40 ft. long, with a capacity of 300 tons of minus 6-in., 4-in., or 3-in. coal per hour. The table may be divided down the center line,

Cleaning results at Eccles, W. Va., are given by Ivan Given[8] in Table 3 for minus 1-in. coal.

The effective removal of dirt from different sizes of Parkgate coal, at Handsworth, Sheffield, England, as reported by Webster[9] is calculated by Mott to the form given in Table 4.

The final clean coal consisted of clean coal from a primary table and coal obtained by re-treating the refuse on a secondary table.

In test 1 the raw coal averaged 14.4 per cent ash and the clean coal 7.0 per cent; the final refuse contained 3.5 per cent of float at 1.6 sp. gr. In test 2 the raw coal was 11.0 per cent ash, the clean coal was 5.8 per cent, and the refuse contained 3.5 per cent of float at 1.60 sp. gr. It is apparent from these tests that when cleaning 2-in. to 0 coal, the table is effective to ⅛ in. in one test and 1/16 in. in the other. Indifferent cleaning was shown for the top size in test 1.

TABLE 3.—*Cleaning Results on Minus 1-inch Coal at Eccles, West Virginia*

Coal	Sink at 1.50 Sp. Gr.		Float at 1.50 Sp. Gr.		Total Ash, Per Cent
	Wt. Per Cent	Ash, Per Cent	Wt. Per Cent	Ash, Per Cent	
Raw coal	7.00	41.02	93.00	4.58	6.94
A clean coal	3.02	39.63	96.98	4.22	5.29
C feed	11.95	46.49	88.05	5.44	10.36
C clean coal	4.93	35.36	95.07	5.05	6.55

A, primary table; C, re-treatment table.
The combined cleaned product analyses from 5.4 to 5.5 per cent ash.

TABLE 4.—*Removal of Dirt from Different Sizes on Peale-Davis Tables*

Size, In.	Percentage Removal of 1.60 Sp. Gr. Sinks	
	Test 1 (1930)	Test 2 (1931)
2–1	73	93
1–¾	95	78
¾–½	85	72
½–¼	89	78
¼–⅛	79	81
⅛–1/16	48	77
1/16–1/25	28	59
1/25–1/60	38	21

Berrisford Small Coal Cleaner (1934).—Three models have been developed for the cleaning of minus 1¼-in. or smaller coal. These cleaners have some unusual design features: (1) the refuse moves uphill in a direction opposite to that of the clean coal, and the upper or refuse end is narrowed, causing concentration of the refuse in a dense mass, which tends to throw out entrapped light coal; (2) only a part of the deck has a porous bottom; (3) a deduster is attached to the table. A single-stage machine is shown in Fig. 23 and one with a deduster in Fig. 24; also, the deflectors at the refuse-discharge end, common to all of these designs, is shown. A two-stage machine equipped with a deduster is shown in Fig. 25. The manufacturers claim that a very clean refuse is obtained and that the cleaning of minus ½-in. slack is effectively accomplished to 60 mesh, and for some coals to 100 mesh.

TABLE 5.—*Comparison of Cleaning Results on Wet Jigs and Air Tables*

Coal	Size, Mm.	Raw Coal	Clean Coal			
			Jig Washer		Air Table	
		Ash, Per Cent	Yield, Per Cent	Ash, Per Cent	Yield, Per Cent	Ash, Per Cent
A	½–10	10.9	88.0	5.8	88.6	7.6
B	½–5	12.5 to 14.0	84.7	5.1	88.8	6.3
B	5–10	15.3	83.7	6.0	82.8	6.4
C	½–10	11.0 to 12.5	86.1	5.2	76.4	6.2
D	½–3	10.8	80.5	6.0	80.4	5.3
D	3–10	9.0 to 9.7	90.3	5.0	83.1	7.7

Bemag-Meguin Process (1929).—The Bemag-Meguin dry-cleaning process was introduced commercially in 1929 by the Bemag-Meguin Co. of Cologne, Germany, although experimental work was carried on during the previous two years. Several plants were constructed using this table. Fig. 26 is a reproduction of Mott's illustration of this table.[4] The table consists of a number of adjustable sections, which together make up a half table. The riffles are inclined downward from the inner to the outer side. The table is reciprocated and air is passed through the perforated deck much the same as in other air tables. Each section of the deck has a three-point suspension, so that it can be adjusted longitudinally or transversely during operation. The air pressure to each of the three sections is adjustable. The capacity is from 14 to 20 tons per hour. A comparison of

cleaning results of the same coal on a Bemag-Meguin jig washer and an experimental Bemag-Meguin air table was given in the *Iron and Coal Trades Review* of Jan. 14, 1929, and is partly reproduced in Table 5.

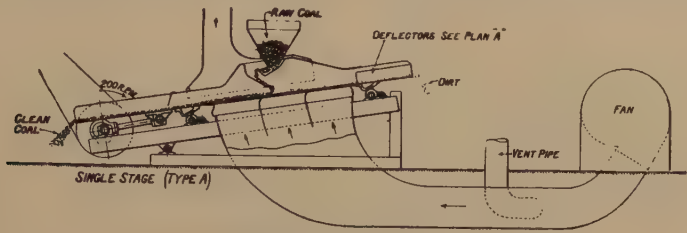

FIG. 23.—BERRISFORD SMALL COAL CLEANER.

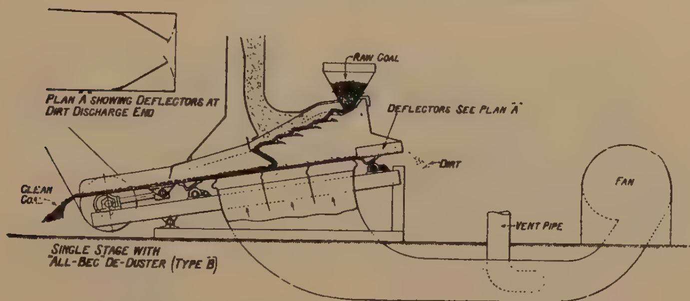

FIG. 24.—BERRISFORD SMALL COAL CLEANER WITH DEDUSTER.

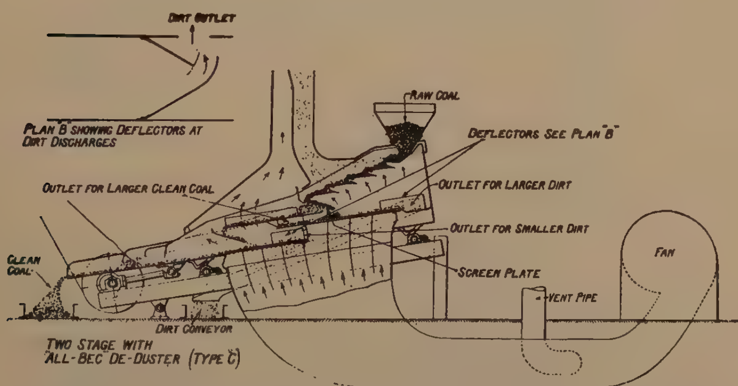

FIG. 25.—TWO-STAGE BERRISFORD SMALL COAL CLEANER WITH DEDUSTER.

KRM Table (1930?).—The KRM table was developed in France by M. Meunier about 1930, and has been rather successful in handling slack coals of high refuse content. It was introduced to the United States in 1936 by the Koppers-Rheolaveur Co., and experimentally tried out on a West Virginia and a Pennsylvania coal. It was found that for efficient operation on coals of the United States containing much less refuse, the table would require considerable redesigning. This was not done, and the table is not on the market at the present time in this country. Over 50 installations have been made in Europe.

The table (Fig. 27) resembles slightly the Bemag-Meguin table, in that there are three sections. The features of this table are: (1) air volume can be closely regulated to each section and to individual parts of

each section, (2) the bed is pinched in at the discharge end of each section, the clean coal discharging over V-shaped riffles into the discharge chute.

matically the deck plan and sectional elevation and Fig. 29 a typical plant arrangement.

The basic features of this process are that

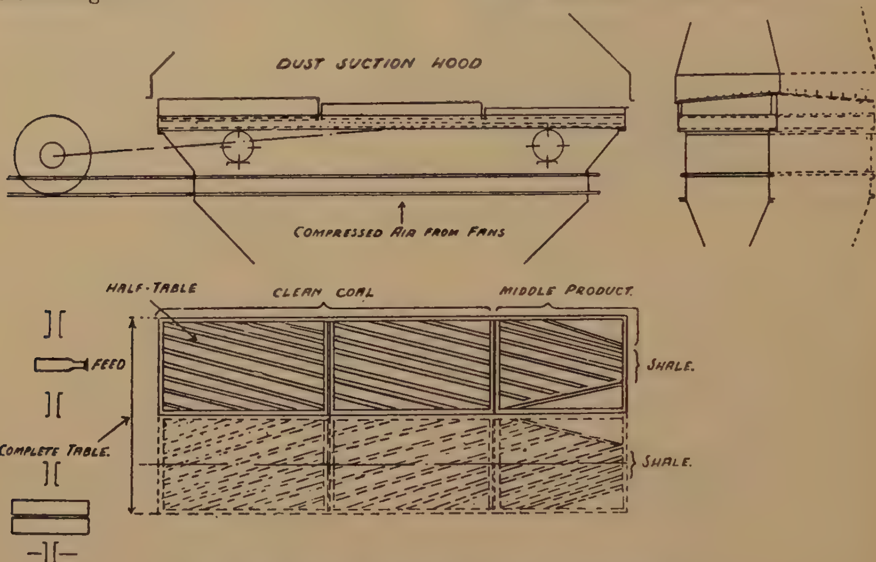

FIG. 26.—BEMAG-MEGUIN COAL-CLEANING TABLE.

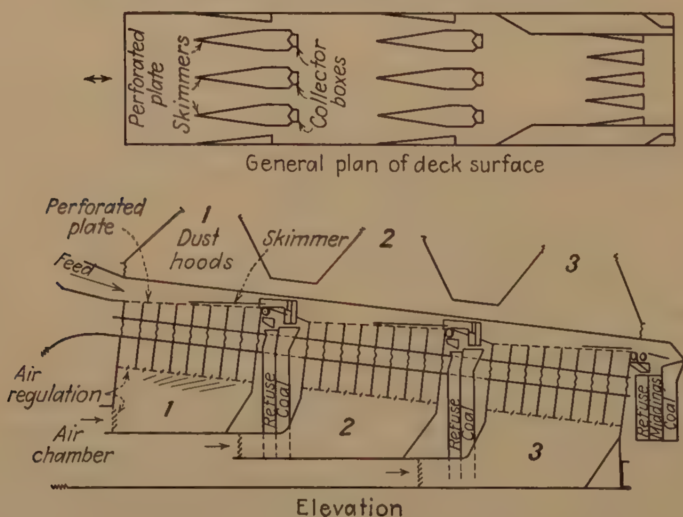

FIG. 27.—KRM TABLE.

Launder-type Cleaners

Static (Raw) Process (1928).—The principles of this process have been recorded by Raw and Ridley.[10] Fig. 28 shows diagram- it consists of subjecting a 5-in. bed of coal in a perforated launder or trough to a pulsating air current while the trough is reciprocated. The trough is built in three or four sections, progressively narrower in

width so that as the clean coal is skimmed off the pinching together of the refuse layer tends to free coal particles. Also, the coal layer is kept at about the same height, so that it can be skimmed off easily. The middling from the last section is usually recirculated. The trough is suspended by flexible steel hangers 5 by ½ in. in cross section.

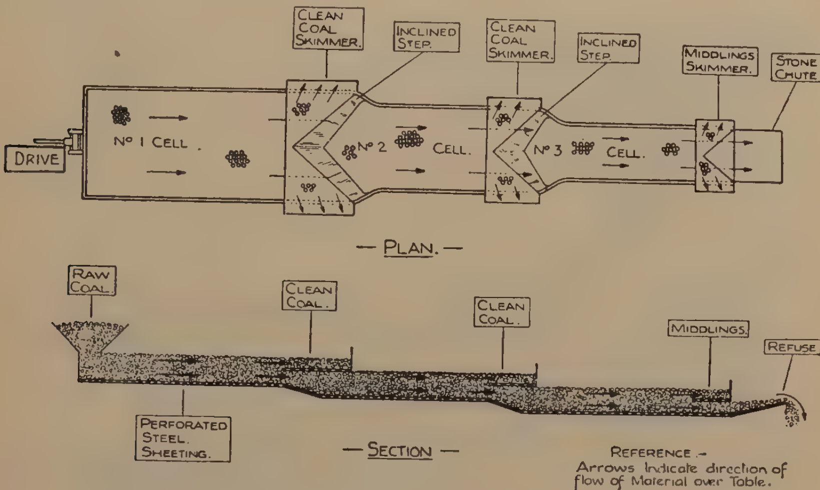

FIG. 28.—DIAGRAMMATIC REPRESENTATION OF DECK AND COAL BED, RAW PROCESS.

ing air pressure gave stratification in one third to one sixth the time of a steady air pressure. The number of pulsations is made about 90 less than the number of reciprocations (400 per min.). The cell lengths of the latest type of four-cell launder are: first cell, 9 ft., 1½ in.; second, 5 ft. 2½ in.; third, 3 ft. 4 in.; fourth, 2 ft. 9 in.; total, 20 ft., 6 inches.

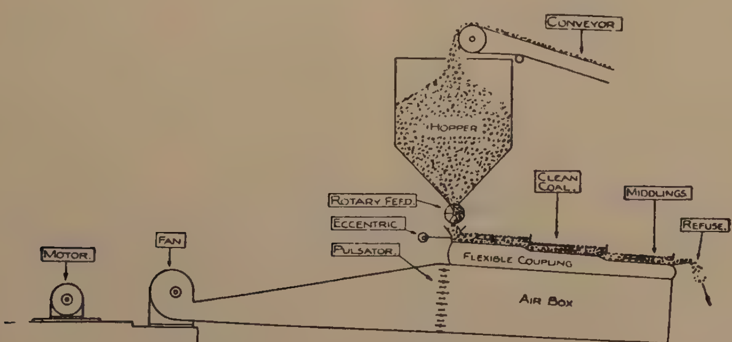

FIG. 29.—DIAGRAMMATIC REPRESENTATION OF RAW PROCESS.

It was found that a critical air pressure equal to 0.67 in. water gauge was necessary for each inch of thickness of the bed to get a fluid condition. Below this value the bed is not fluid, while above it tends to "boil." It was also found that an oscillating or pulsat-

The bottom of the trough is perforated plate (1⁄12-in. holes; 50 per cent air space) reciprocated by a plain eccentric at ½-in. stroke. In the three-cell launder, the first cell is 3 ft. wide, the second, 2½ ft. wide, and the third, 1 ft. 5 in. wide. Inclined

skimmers remove the coal from each cell. Unsized coal from 1½- to 2-in. top size is treated in these separators to ⅛ or 1/16 in. with excellent results. Table 6 gives the results of tests made on ¼-in. to 0 coal from the Yard seam, Northumberland.

It is to be noted that the cleaning efficiency, as indicated by the percentage of sinks in the clean coal, falls off below 1/16 in. in size, and in general it can be said that this cleaner is good down to ¼ in., reasonably good down to ⅛ in., but is only fair from ⅛ in. to 1/16 in. and falls off altogether below this figure. Accordingly, if a coal contains a very large percentage of dust below 1/16 in. in size, this dust may have a pronounced effect on the final ash content of the total clean coal. In normal circumstances, average north of England coals are being cleaned on the static cleaner to within 2 per cent of their theoretical ash content as shown by a washability curve, taking the whole coal from, say, 1 in. to 0 or 2 in. to 0.

Mott[11] records results obtained at Thornley colliery, England, where two four-cell launders are cleaning 100 tons per hour of 2-in. to 0 coal (Table 7). The two tables at this plant are arranged so that their air ducts are connected to one fan. When the pulsator of one air box is open, the other is closed. One fan thus serves to produce the static pressure for two tables. The middling skimmed from the fourth cell is recirculated.

This cleaning plant is operated by two men. The power requirements are 170 hp. and the total operating costs 1.9d per ton. The capital cost of the plant, steel and brick construction, was approximately 15,000 pounds.

Barker Coal Cleaner (1935?).—This launder type of cleaner introduced about 1935 resembles somewhat the static cleaner in general features. A photograph of the unit is shown in Fig. 30. It consists of a steel trough having a perforated deck, progres-

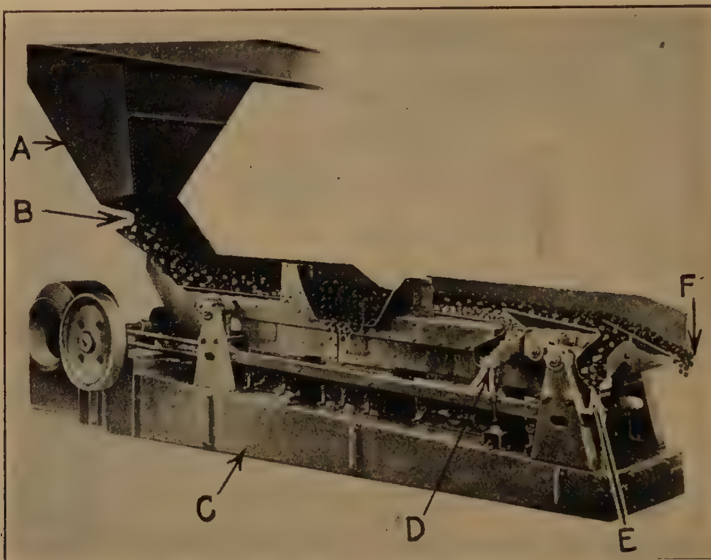

Fig. 30.—The Barker coal cleaner.
A, surge hopper with automatic control.
B, raw coal.
C, air box.
D, dirt.
E, middlings.
F, clean coal.

sively narrowed to get three sections of varying width. Essential features of this machine are: (1) a continuous air current is used; (2) the launder is mounted on specially designed bell-crank levers, giving a nearly vertical motion, which tends to prevent uneven air flow through the 4 to 5-in. bed of coal in the trough; (3) no skimming plates are used, the upper layer

TABLE 6.—*Coal Tested on Static Cleaner*
YARD SEAM, NORTHUMBERLAND, JUNE 1933
Test of 1/4-inch to 0 Coal

Inch	Weight, Per Cent	Ash, Per Cent	Float in 1.6 Sp. Gr.		Sink in 1.6 Sp. Gr.	
			Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
RAW COAL						
1/4–1/16	55.7	17.6	77.1	3.2	22.9	66.1
1/16–1/32	19.7	18.6	75.5		24.5	
1/32–1/50	7.2	21.9	71.0	3.1	29.0	67.8
1/50–1/64	2.4	19.9	74.0		26.0	
1/64–0	15.0	20.3				
3/4–0	100.0	18.6				
CLEAN COAL						
1/4–1/16	58.0	4.9	95.9	3.2	4.1	45.3
1/16–1/32	22.0	8.9	86.5		13.5	
1/32–1/50	8.6	19.5	74.0	2.9	26.0	67.1
1/50–1/64	1.9	20.9	72.0		28.0	
1/64–0	9.5	25.0				
1/4–0	100.0	9.2				
1/4–1/32	80.0	6.0				
1/32–0	20.0	22.2				

Shale: Ash, 75.2 per cent. Free coal (floats in 1.4 sp. gr.) = 1.2 per cent.

of coal simply spilling out over the sides as the launder is narrowed.

The capacity of a unit 2 ft. wide is approximately 25 tons per hour. It is understood that several of these machines have been installed, but no operating data are available.

DENSE-MEDIUM PROCESSES

Air-sand Process (1942?).—The idea of making a dry, specific-gravity separation of coal from refuse by utilizing an artificial dense medium intermediate in specific gravity between refuse and coal is an attractive one. The only process to achieve commercial success is Fraser's air-sand process, marketed by the Stephens-Adamson Mfg. Co. The dense medium is formed by bubbling air through a mass of dry sand,

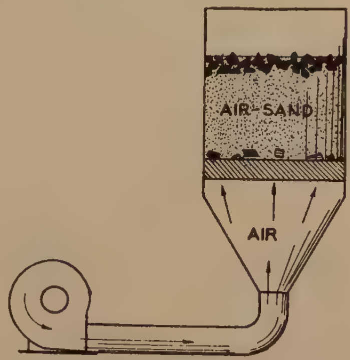

FIG. 31.—FRASER'S AIR-SAND PROCESS.

thereby aerating it and causing it to behave as a heavy liquid, the coal floating on the aerated sand mass and the refuse sinking (Fig. 31).

TABLE 7.—*Cleaning Results at Thornley Colliery*

Size, In.	Raw Coal		Clean Coal	
	Weight, Per Cent	Ash, Per Cent	Weight, Per Cent	Ash, Per Cent
2–1/4	52.5	10.7	41.2	4.2
1/4–1/16	29.7	11.2	34.9	5.2
–1/16	17.8	15.8	23.9	11.0
.2–0	100.0	11.8	100.0	6.0

A diagrammatic section of a three-cell machine is shown in Fig. 32. This process has been installed at several plants and has proved to be efficient and economical in cleaning the coarser sizes of coal down to 1/4 in. en masse. Theoretically there is no limit to the size ratio that can be handled. The chief limitation of this process is that sizes smaller than about 1/8 in. cannot be cleaned. The Stephens-Adamson Mfg. Co. also markets a fine-coal cleaner, similar in principle to the air-sand, for cleaning coal from 3/8 in. to 0.

The Prins Process (1933?).—The Prins process (Fig. 33) is described in U. S. Patent 431801. It was tried for a time at one of the mines of the Peabody Coal Co. in Illinois. is screened from the coarse refuse and returned to the head of the trough by a conveyor and elevator. The floating large coal passes over skimmers in the trough to

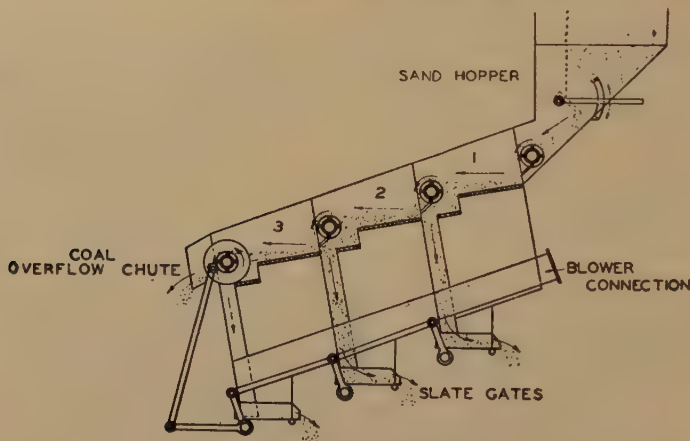

FIG. 32.—THREE-STAGE AIR-SAND SEPARATOR BOX.

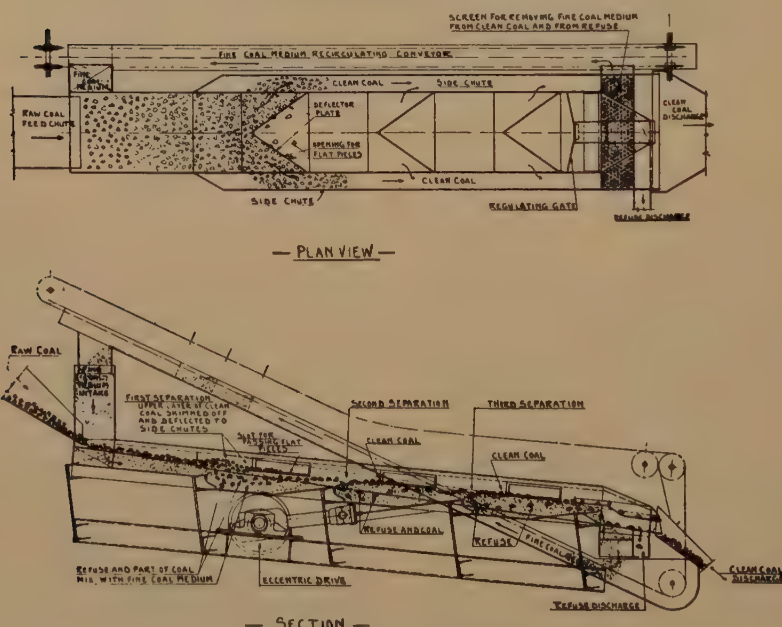

FIG. 33.—THE PRINS PROCESS.

In this process large-size coal is separated from refuse in a flowing bed of small coal in a reciprocating troughlike separator. The refuse sinks to the bottom. The small coal the discharge chute. Cleaning results at the Illinois mine where this process was tried gave fair results on the clean coal, but the refuse contained a large proportion of coal.

References

1. *Mechannual* (1941) 140.
2. W. H. and S. R. Berrisford: The Dry Separation of Small Coal. *Trans.* Inst. Min. Engrs. (1926) **72,** 97.
3. R. R. Slaymaker: Development of the Air-Float Method of Gravity Separation. *Trans.* Amer. Soc. Mech. Engrs. (1936) **58,** 443.
4. R. A. Mott: Recent Developments in the Dry Cleaning of Coal. *Trans.* Inst. Min. Engrs. (1931) **81,** 139.
5. *Colliery Engineering* (1931) **8,** 451.
6. D. H. Davis and V. D. Hanson: Some Factors Affecting Results when Air Cleaning Fine Coal. Amer. Min. Congress *Jnl.* (May 1939) **25,** 10.
7. K. C. Appleyard: Dry Cleaning of Coal in England. *Trans.* A.I.M.E. (1931) **94,** 235.
8. I. A. Given: Mechanical Cleaning—Puts Crab Orchard Slack in By-Product Class. *Coal Age* (1932) **37,** 294.
9. N. E. Webster: Notes on Washing vs. Dry Cleaning (Peale-Davis Method) of Parkgate Coal. *Trans.* Inst. Min. Engrs. (1930) **81,** 175.
10. G. Raw and F. F. Ridley: The Principles of Pneumatic Separation with a Description of the Static Dry Washer. *Trans.* World Power Fuel Conference (1928) **1,** 385.
11. R. A. Mott: Dry Cleaning at Thornley Colliery. *Colliery Engineering* (July 1934).

Control of Solids in a Closed Washery Water System

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(New York Meeting, February 1941)

Comparatively little has been published relating practical experience with the control of the solid content of washery water systems that must be "closed." A "closed" system is one that for some reason may not be cleaned by the emission of any effluent of high solid content to be replaced by clean water in order to hold the solid content of the waste water within reasonable bounds for efficient washing. It is the purpose of this paper to describe practical experience with such a water system and to present data that it is hoped may be of some use to those who are faced with a similar problem.

The washery clarification system described comprises three thickeners and three filters, and although it is admitted that each of the fields of water settling and thickening and filtration offers a wide scope for investigation, the authors are not attempting to present the answers to these problems but to set down the results of some experience along these lines, making no claims as to the ultimate solution of the problems in hand. The subject of water clarification has vast ramifications; for example, the science of flocculation as applied to water settling and filtering opens up fields of wide possibilities of which as yet little is known. A comparatively recent entry into the field of water treatment, which may offer possibilities of practical application, is the use of "wetting" agents such as "Aerosol."

General Arrangement of Water System

So that the reader may obtain a comprehensive picture of the factors concerned in the control of a closed water system, it is necessary to give a brief description of the washing plant that is serviced by the water system to be described in the following pages.

As we are not primarily concerned with the washing plant in detail, only a brief description of the flow of coal as it relates to the water system will be given here. For a more detailed account of this type of washer, the reader is referred to earlier papers.[1,2]

The washer, which handles 0 to 4-in. raw coal (750 tons per hour) is an American Rheolaveur plant of two complements; the coarse coal or sealed discharge plant and the fine coal or free discharge plant. The sealed discharge plant (Fig. 1) consists of twin 48-in. primary launders, each having two washing boxes, supplemented by one 32-in. rewash launder, which also has two washing boxes. "Push" water for washing and conveying coal in these launders is supplied from a constant-head tank under pressure head of 10½ ft. at the point of introduction at the feed end of the launder. This tank also supplies water for upward or vertical currents used in the Rheolaveur or washing boxes. Each box discharges into a sealed boot elevator (Fig. 1).

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[1] References are at the end of the paper.

The feed, consisting of 750 tons per hour of minus 4-in. raw coal, enters the upper end of each primary launder, where it is met by the "push" water, which, in conjunction with the slope of the launder, conveys the coal down the launder. It passes over No. 2 box, which takes out primary refuse and thence over No. 1 box, where middlings material is removed, and finally overflows the end of the launder as clean coal. The middlings are elevated to the head end of the launder and sluiced back in with the feed coal. The primary refuse is elevated to the head end of the rewash launder.

Washing is done here in much the same manner as in the primary launders, the feed passing over No. 4 box, which discharges final refuse, and thence over No. 3 box, which discharges a middling recirculation material, and finally overflowing the end as clean coal. Vertical currents are used in all boxes but with varying degree, most being used in the No. 4 (final refuse) box and least in the No. 1 (primary middlings) box.

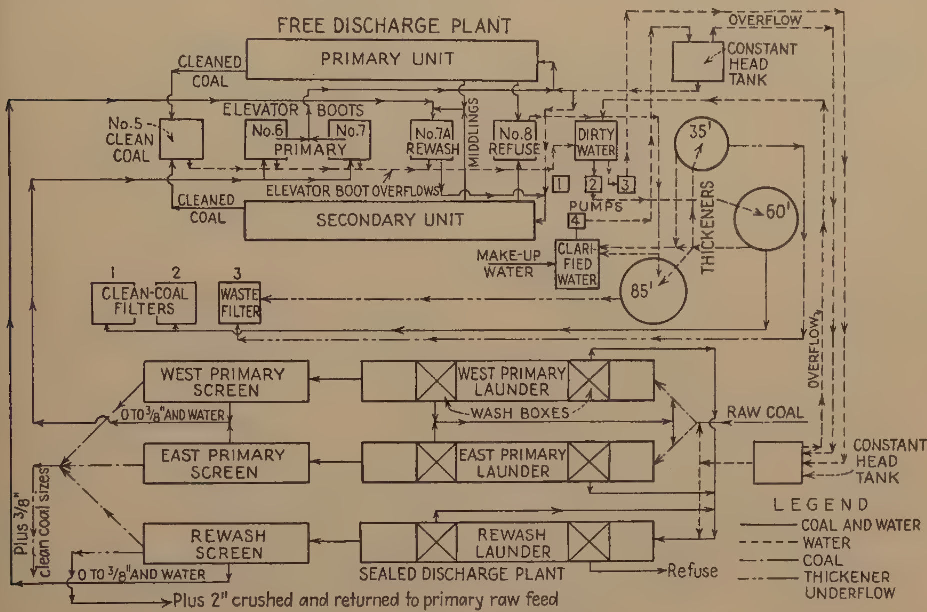

FIG. 1.—WASHERY FLOWSHEET.

The clean coal and water overflowing each launder discharges to its own bank of reciprocating screens, which both size and dewater. The screens are of the Parrish type, consisting of three decks that size at 2-in., 1¼-in. and ⅜-in. mesh, respectively Each bank of screens is divided into two sections, the upper and lower, which are balanced and driven from a single crankshaft.

The water and fine coal passing through the bottom (⅜-in. mesh) screen deck are sluiced to the feed elevator boots of the free discharge plant. The materials through the ⅜-in. screen deck of both primary screens go to No. 6 and No. 7 elevator boots, which feed one side of the free discharge plant (Fig. 1). The materials passing through the corresponding deck in the rewash screen go to No. 7*A* elevator boot, which feeds the other side of the free discharge plant.

The free discharge plant (Fig. 1) consists of twin units of four launders each, the launders being tiered so that the washing

boxes in *A* launder discharge into *B*, those in *B* discharge into *C*, and so on.

The overflows from *A*, *B* and *C* launders of both units are sluiced to the clean-coal

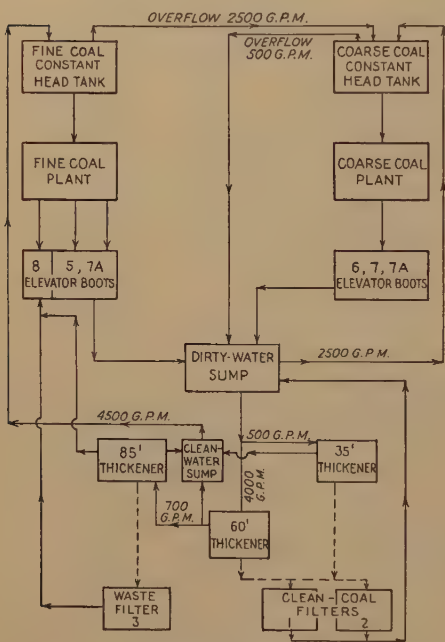

FIG. 2.—FLOWSHEET OF WATER SYSTEM.

elevator boot (No. 5). The overflows from both *D* launders are sluiced to No. 7*A* elevator boot and are rewashed in the secondary side. The materials from refuse boxes, discharging from *C* and *D* launders, are sluiced to No. 8 elevator boot, from which they are elevated and discharged to a conveyor.

"Push" water for washing and conveying the fine coal in the launders, as well as vertical or countercurrent water for some refuse boxes, is furnished from a constant-head tank under pressure head of 18½ ft. at the point of introduction to the *A* launders.

In the foregoing description the reader has followed the flow of water from the constant-head supply tanks through the flow of coal to the free discharge elevator boots. It is at this point, where the water overflows the boots, that the problem of controlling the density or solid content of the wash water begins. Fortunate indeed is the plant operator whose boots are large enough to permit sufficient settling so that the overflow water contains not over 5 or 6 per cent solids—with reservations, of course, in accordance with the type of these solids; that is, the size, shape and specific gravity, percentage of fine slimes, clay and colloidal materials present.

Layout of Water System

The general arrangement of the water system is outlined in Fig. 2. With respect to the solid materials recovered from the water, there are two divisions of the flow of water; namely, one in which a part of the water is returned unclarified to the coarse-coal plant constant-head tanks, and the other in which the remainder of the water is clarified in two divisions, one that carries solids that are settled and recovered as clean coal and the other from which solids are settled and discarded as refuse. This scheme uses two of the thickeners (35-ft. and 60-ft. in Fig. 2) somewhat in the capacity of classifiers, or rather as selective settlers, and the other as a water clarifier (85-ft. in Fig. 2), settling slimes or extremely fine material of which a high percentage is clay.

Table 1 gives the solid content and size characteristics of elevator-boot overflows, thickener feeds, overflows and underflows, as well as gallonages and tonnages of solids at various points. Overflows from all coal boots (5, 6, 7 and 7*A*, Fig. 2) ranging in solid content from 4.9 to 8.7 per cent and averaging between 7 and 8 per cent empty into the dirty-water sump.

Here the boot overflows are joined by the overflow from the sealed discharge constant-head tank, which in turn has taken the overflow from the free discharge constant-head tank. These tanks overflow because, as a safety factor to take care of

sudden increase in water demand due to surge in feed, etc., it is necessary to pump more water than actually is needed in washing coal. In actual gallonages, 2000 gal. per min. is required for the free discharge plant and 4500 gal. per min. for the sealed discharge plant. It should be understood that this total of 6500 gal. per min. of circulating water at 7 to 8 per cent solids need not all be clarified, since experience has shown that in this type of washer better all-round washing efficiency is attained when the water carries about 5 per cent solids, particularly in the fine-coal plant, than when the solids are either appreciably above or below this figure. It is necessary therefore to clarify only enough water for mixing with that of 7 to 8 per cent solid

TABLE 1.—*Sieve Analysis on Dorr-thickener Circuits and Elevator-boot Overflows*

Mesh	No. 5 Elevator		No. 6 Elevator		No. 7 Elevator		No. 7*A* Elevator		No. 8 Elevator		Dirty Water		Clarified Water	
	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash
Head		21.1		20.0		19.9		20.7		24.0		21.8		21.7
+ 48	3.0	6.7	5.0	7.3	2.5	9.0			1.5	13.9	1.0	8.5		
100	9.0	8.4	8.0	7.4	12.0	7.1	6.0	6.1	1.0	18.5	4.0	12.6	2.0	6.5
200	11.5	11.1	11.0	12.5	19.0	13.2	14.5	10.8	6.5	29.4	8.0	15.5	6.5	6.6
−200	76.5	24.5	76.0	23.4	66.5	24.2	79.5	23.5	91.0	24.3	87.0	23.9	91.5	23.3
Percentage of solids	4.9		8.7		7.9		8.7		3.5		6.8		4.0	

DORR THICKENERS

Mesh	Feed, Per Cent Wt.	Ash	60 and 35-ft. Dorr Thickeners Operating in Parallel: 60-ft. Overflow		60-ft. Underflow		35-ft. Overflow		35-ft. Underflow		85-ft. Dorr Thickener: Feed		Overflow		Underflow	
			Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash	Per Cent Wt.	Ash
Head		20.5		22.2		12.2		24.2		16.2		22.6				23.9
+ 48	2.0	6.4			12.0	5.0			5.0	5.2	Tr.	13.9				
100	8.0	6.5	1.5	14.6	40.5	6.6	2.0	15.5	19.0	6.1	1.5	15.2	Clear		1.0	12.3
200	14.0	12.5	11.5	14.0	22.5	12.7	4.5	21.9	22.0	9.1	10.5	15.7			8.0	18.3
−200	76.0	24.0	87.0	23.7	25.0	29.5	93.5	24.2	54.0	24.8	88.0	23.8			91.0	24.5
Percentage of Solids	6.6		4.7		32.9		3.9		36.2		4.5				35.3	
Gal. per Min.	4,500		3860		140		457		43		850				64	
Tons per Hour	74.3		45.4		12.5		4.5		4.3		9.5				6.2	

CLEAN-COAL FILTER CAKE

Mesh	Feed, Per Cent Wt.	Ash
Head		14.6
+ 48	9.0	5.0
100	33.0	6.5
200	26.0	12.3
−200	32.0	27.6

Clarification, 85-ft. thickener, 20 inches.
Coagulents, 85-ft. thickener: 3.8 lb. potato starch and 1.9 lb. caustic soda per operating hour.

content to maintain a balance of 5 per cent solids in the water available for washing.

As mentioned before, 6500 gal. per min. of water at 7 to 8 per cent solids runs into the dirty-water sump from free discharge and sealed discharge plants. For the type of settling desired in the 60-ft. and 35-ft. thickeners, 4500 gal. per min. must be pumped to the 35-ft. and 60-ft. thickeners in order to obtain selective settling; that is, allowing as little high-ash minus 200-mesh to settle as possible but at the same time avoiding an overflow of the coarse lower-ash plus 200-mesh materials back into the water system.

As shown in Fig. 2, this feed is split, 4000 gal. going to the 60-ft. and 500 going to the 35-ft. thickeners, and, instead of carrying from 7 to 8 per cent solids, has dropped to 6.6 per cent solids. This is caused by dilution with excess pumpage of clarified water, consisting of approximately 500 gal. per min. of combined thickener overflows at about 3.5 per cent solids, which constitutes the overflows to the dirty-water sump from the sealed discharge and free discharge constant-head tanks. No. 2 pump is used for feeding the thickeners and No. 1 or No. 3 pump may be used for handling the remainder of the water in the dirty-water sump. One pump only is needed for this purpose, holding the other as a spare.

Now, collecting a few loose ends left here and there out of 7000 gal. per min. coming into the dirty-water sump (6500 from washing plants and 500 recirculation or excess) we have pumped 4500 to the thickeners, leaving 2500 gal. per min. to be pumped unclarified directly to the sealed discharge constant-head tank.

Leaving that for the moment, let us follow the 4500 gal. per min. through the thickeners. Of the latter, 500 gal. per min. is fed to the 35-ft. thickener at 6.6 per cent solids (Table 1) having a head ash of 20.5 per cent and being 76 per cent minus 200-mesh, which has an ash content of 24.0 per cent. The underflow has dropped in ash content (16.2 per cent) and correspondingly in proportion of minus 200-mesh (54.0 per cent). The overflow has decreased in total solids from 6.6 to 3.9 per cent and increased in ash content (24.2 per cent) while the percentage of minus 200-mesh has increased to 93.5 per cent. The underflow is pumped to the coal filters and the overflow joins that of the 60-ft. and 85-ft. thickeners.

Next, considering the 4000 gal. per min. feed to the 60-ft. thickener, which, of course, has the same characteristics as those of the 35-ft. thickener feed, it will be noted that much better differential settling is being done. The underflow has dropped to 12.2 per cent head ash and the proportion of high-ash minus 200-mesh therein has decreased to 25.0 per cent. The overflow has been reduced to 4.7 per cent solids while the head ash has increased to 22.2 per cent and the proportion of minus 200-mesh has increased to 87.0 per cent.

It is interesting to note that, in trying to eliminate the high-ash minus 200-mesh from the clean-coal underflow, although the quantity of this size has been considerably decreased, at the same time the ash content in the fraction of this size that settles with the coarser sizes has been increased. This is natural, since in differential settling the lighter, lower-ash particles of a given size will be buoyed up while the heavier higher-ash particles will sink. In apparent contradiction to this theory, however, in the 48 to 100 and 100 to 200-mesh sizes the ash is considerably *higher* in the *overflow* than in the corresponding sizes of *underflow*. This may be explained as due to the flaky structure and flat shape of the high-ash impurities in these sizes, characteristics that cause the particle to be more affected by the upward pressure of the overflow than by their specific gravity.

It is evident, from the data examined up to this point, that while the proportion of

slimes in the coal sludge has been reduced, the proportion of slimes in the circulating water is tending toward increase. Experience has shown not only that this is true but also that as the proportion of slimes increase the problem of water settling becomes more difficult. For this reason it is necessary to subject a part of the 60-ft. thickener overflow to further settling treatment, and this is done in the 85-ft. thickener.

Approximately 700 gal. per min. of the 60-ft. overflow, together with 150 gal. per min. overflow from the refuse boot (No. 8), which is fed directly to the 85-ft. thickener because of the high ash characteristics of its solids and also because of its high slime content (minus 200 mesh), constitute the feed to the 85-ft. thickener. The result is a very difficult water-clarification problem.

It was found that by natural settling alone it was impossible to maintain the water system in balance with the settling capacity available and operate with a closed system. Beginning with a clean system at the first of the week, the percentage of solids in the circulating water gradually built up to 12 or 13 per cent at the end of the week, despite the fact that underflow from the 85-ft. thickener was wasted during operating hours and usually for 1 to 1½ hr. before the plant began operating each day. In addition, it was necessary to pump out the 85-ft. thickener over weekends, to get rid of accumulated solids that settled only when the thickener was not in operation. These troubles had the attendant evils of expense for cleaning the sludge pit, loss of purchased water, extra pumping expense and a loss of washing efficiency, particularly in the fine-coal plant, through "heavy" water.

After a lengthy and thorough research into flocculating agents as an aid to the settling of solids in water, the use of causticized starch in the 85-ft. feed was finally adopted, with very good results. A clear overflow is now being maintained from the 85-ft. thickener and the wasting of underflow to the sludge pits has been almost discontinued. All underflows are handled by filters and the filtrate water is returned to the system. The system is held in balance at all times during the week; that is to say, the solid content of the wash water is held between 5 and 6 per cent. The amount of causticized starch necessary to produce this gratifying change, which virtually triples settling capacity, is about 23 lb. per 7-hr. shift added to the 85-ft. thickener feed, or about ½ lb. per ton of solids settled on a dry basis.

It is not within the scope of this paper to present a discourse on the theory or mechanics of settling and flocculating agents. Gardner and Ray[3] performed most of their experiments on the 85-ft. thickener feed and have included in their paper a rather complete bibliography on the subject. It is interesting to note that at the time these investigators were making their study the feed to the 85-ft. thickener contained only 64 per cent minus 200-mesh and that since the introduction of causticized starch this figure has increased to 93.5 per cent, indicating a general cleaning up of the system.

Completing the circuit of the water system, the overflow of the 35-ft. thickener, being 460 gal. per min. at 3.9 per cent solids, plus that of the 60-ft. thickener, 3010 gal. per min. at 4.7 per cent solids, together with the overflow of the 85-ft. thickener, 790 gal. per min. at 0.0 per cent solids, are all returned to a common sump. From this No. 4 pump takes its full capacity of 4500 gal. per min. of discharge water at about 3.5 per cent solids to the free discharge constant-head tank. The remainder of the water from the dirty-water sump is taken by No. 1 pump (approximately 2500 gal. per min.) and is discharged to the sealed discharge constant-head tank.

Make-up water is added to the outside or thickener overflow sumps at the rate of approximately 100 gal. per min., to replace

water lost: (1) by evaporation of moisture from 0 to 3/8-in. material in the heat drying plant, (2) moisture in clean 3/8 to 4-in. sizes left after drainage on booms and in cars and (3) moisture lost in refuse from coarse-coal and fine-coal plants and from No. 3 filter.

Test data of solids in a water system of this size invariably show discrepancies, particularly on tonnage balances, unless the data are obtained in sampling periods covering a long operating time that embraces sufficient shutdown and operating periods to give average over-all conditions. Such accuracy is not necessary for operating control, and the data given are largely gleaned from tests ranging over 2 to 4-hr. operating periods. For this reason tonnage figures of feed, overflow and underflow of thickeners do not usually balance out within 2 or 3 tons per hour, and these results must be interpreted rather broadly to obtain the correct picture.

Within recent months, since the collection of data for this paper, the water system has been undergoing a constant change, due principally to the large increase in mechanically loaded tonnage input and its attendant increase in quantity of clayey refuse or draw slate in the feed to the washer. Because of rapid slaking of the draw slate in transit and in the water at the Champion No. 1 plant, the increase in tonnage of the material has brought increased difficulty in settling, particularly in the 85-ft. thickener and in filtering the 85-ft. thickener underflow. Increasing the quantity of causticized starch has been of little benefit and the use of lime with the reagent has helped little in settling the extremely fine and colloidal clayey material. The use of dry lime with the 85-ft. thickener underflow, however, has proved to be of great value in conditioning the underflow for filtering, and this phase of water-system control will now be discussed briefly.

TABLE 2.—*Comparison of Clean Coal and Refuse Filter Products Obtained from Selective Setting*

Material	Dry Solids, Tons per Hr.	Moisture in Cake, Per Cent	Cake Thickness, In.	Sieve Analysis of Cake: Head Ash	+48 Mesh		48 to 100 M.		100 to 200 M.		−200 Mesh	
					Wt.	Ash	Wt.	Ash	Wt.	Ash	Wt.	Ash
Clean coal	16.6	22.3	3/8	12.5	6.0	4.7	31.0	6.1	29.0	10.7	34.0	23.2
Refuse	9.1	35.2	1/4	30.5	2.0	14.8	7.5	18.3	15.0	24.5	75.5	33.1

EFFECT OF SELECTIVE SETTLING ON FILTERING

Generally, it may be said that the filtering operation is performed most easily on 48 to 200-mesh material in regard to water removal, maximum tonnage of filter cake, minimum moisture content of the discharged cake, and freedom of effluent from solid content. As the amount of minus 200-mesh increases beyond 20 to 30 per cent, the tonnage drops and moisture in the filter cake increases.

As the character of the minus 200-mesh material changes from granular to colloidal or slimy, the filtering problem becomes increasingly difficult.

Table 2 illustrates the difference in characteristics of the sludge products resulting from differential settling. The reader's attention is directed to the higher moisture and ash content, and the higher percentage of minus 200-mesh material of the refuse filter product, as compared with similar characteristics of the clean-coal filter product.

Summary

It has been the aim of the authors to present some practical discussion of water-system problems rather than voluminous data with scientific interpretation. Primarily a method of "selective" settling that aids in coal recovery has been discussed. The use of lime and causticized starch as filtering aids has proved of considerable value but there is a definite need for more efficient and cheaper agents in the field of slime settling and filtering as related to coal-washing practice.

By means of "selective" settling, approximately two thirds of the sludge from the washery is recovered in such a form that it can be more economically conditioned for loading as a finished product, while the remaining one third is of such poor quality that at present it is discarded as refuse. The refuse contains such a large percentage of fine clay that at present it is not economical to attempt the recovery of the fine coal associated with it. The quantity of this material wasted as refuse roughly approximates the quantity of degradation of similar size consist at a central cleaning plant of this type, resulting from the loading, transporting, dumping and cleaning of the raw coal and the loading of the finished product.

Acknowledgments

Many thanks are due to Mr. J. B. Morrow, President of the Pittsburgh Coal Co., for permission to use the data and to relate operating experience gained in the plants that were being operated under his immediate direction.

References

1. J. T. Crawford, C. P. Proctor and M. J. Williams: Launder Washing of Coarse Coal. *Trans.* A.I.M.E. (1938) **130**, 172.
2. J. T. Crawford, C. P. Proctor and J. A. Younkins: Launder and Table Washing of Fine Coal. *Trans.* A.I.M.E. (1940) **139**, 269.
3. G. R. Gardner and K. B. Ray: Flocculation and Clarification of Slurries with Organic Flocculants. *Trans.* A.I.M.E. (1940) **139**, 290.

A New Graphic Presentation of Coal-cleaning Characteristics

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In the presentation which follows, washability curves, such as are commonly used in making studies preliminary to the cleaning of any coal or to the concentration of any mineral, have been reduced in each instance to a single straight line; thus all the most important characteristics of the coal or mineral are represented in a form more advantageous than that in which such data usually are shown.

To obtain consistent results, float-and-sink data must be based on a single definite narrow size range, which may be designated "a coal unit." It will be shown that such units, each embodying a different size range, can be grouped to form what may be termed "a composite coal." A three-dimensional presentation will emerge, which will closely identify the coal, illustrate its characteristics in a new manner and explain many obscure facts.

Float-and-sink Tests

For the preliminary study of any cleaning or concentration problem, or in the control of any operation, float-and-sink tests must be made. The results of these tests are grouped in tabular form and then illustrated in various systems of curves; but if such a graphic representation is to have the utmost practical value, it must offer at least the following advantages: (1) It must be easier to read than the tabular results, and (2) it must be easy to plot, check and interpolate.

The first requirement will be attained by reducing the number of curves to a minimum, and the second by some system of representation that will use only straight lines. Of the many washing curves now in use, only the "recovery curve," often known as the "composite curve"—a somewhat misleading term—will be considered.

Any point on the recovery curve will show what percentage of weight of clean coal can be recovered when the mineral is washed at a certain specific gravity. As an example, the tests listed below Fig. 1 are illustrated in the drawing by a curve obtained by plotting (1) as abscissas the ash content of the coal recovered and (2) as ordinates the corresponding percentage of refuse discarded; but, in general practice, instead of writing down that weight, it is customary to set down its complement, which is the corresponding weight of coal recovered. From this curve can be deduced the information necessary for the solution of most practical problems.

New Form of Graphic Presentation

The resemblance of the recovery curve in Fig. 1, or, for that matter, the recovery curve obtained in common practice, to a power curve is apparent. The general equation of such a curve is $y = ax^n$, where n is negative.

Taking logarithms: $\log y = a + n \log x$. Using a logarithmic scale, the equation of the recovery curve will be $Y = A + bX$.

On this logarithmic basis, the recovery graph will be a straight line, using the same variables as in Fig. 1; namely, the ash con-

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tent of the coal recovered as abscissas, and the weight of refuse discarded, or its complement (weight of coal recovered), as ordinates.

Any standard double-logarithmic paper can be used, and if none is available one can be made with the aid of a slide rule. Such multiples may be selected as will best fit any particular problem, and they thus will enlarge, as with a magnifying glass, the zone of practical interest. To illustrate this system, data will be taken from previously published technical papers, and from the *Bulletins* and *Reports of Investigations* of the U.S. Bureau of Mines.

Plotting Unit Washability Graphs

"A unit washability graph" illustrates the results of float-and-sink tests as applied to a definite size range. A sample of coal from No. 3 bed in Indiana, size 0 to ¼ in., as reported in U.S. Bureau of Mines *Bull.* 300, page 109, serves as an example. It has an ash content of 16.2 per cent.

Table 1 gives successive recoveries of cleaned coal expressed in percentage weights of the original sample. These are plotted in Fig. 2 on the vertical axes OY; on the horizontal axis are plotted successively, each on a separate scale and starting from a separate origin: (1) the percentage ash content of the cleaned coal; scale line OX; (2) the specific gravity of separation; scale line GG; (3) the sulphur content of the cleaned coal; scale line SS; and (4) the percentage ash content of the refuse; scale line R_aR_a.

For further convenience, the vertical line on the right, XZ, may be graduated so as to be readable directly in percentage weight of refuse, complemental weight of the cleaned coal.

To illustrate possible uses of these graphs, they may be employed to ascertain the results obtained when washing at, say, a specific gravity of 1.55. From the point g on the scale of specific gravities GG, a vertical line is dropped, which intersects G_1G_1 at point a. The horizontal line from a intersects to the right the recovery graph C_1C_1 at b; refuse-ash graph R_1R_1 at c; and on the left, the sulphur graph S_1S_1 at d.

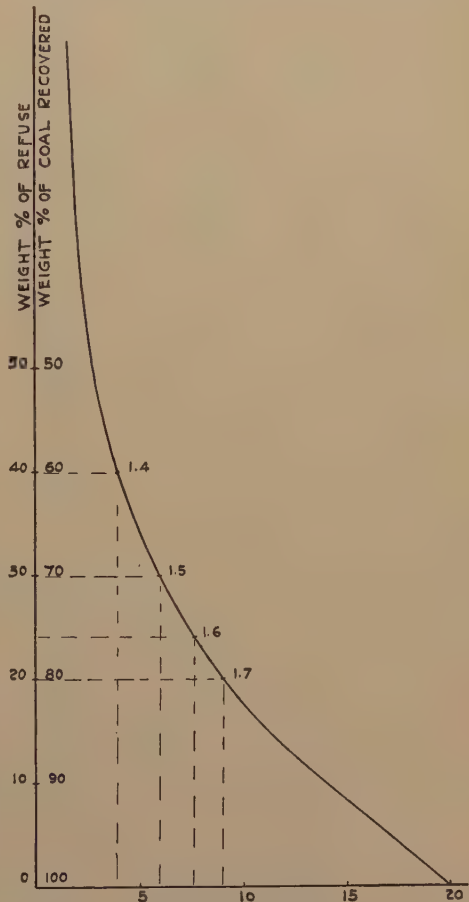

Fig. 1.—Recovery curve.

Sp. Gr.	Wt. Per Cent Coal Recovered	Ash Content, Per Cent
1.4	60	4.0
1.5	70	6.0
1.6	76	7.5
1.7	80	9.0
	100	20.0

The exterior vertical lines showing percentage weight of cleaned coal and percentage weight of refuse are cut, respectively, at 85.5 and 14.5. The points b, c and d, when read on their respective scales, show values

TABLE 1.—*Float-and-sink Washing Test*
SIZE 1 TO ¼ INCH, NO. 3 BED, INDIANA

Scales of Ordinates			Scales of Abscissas			
Per Cent Weight of Refuse	Per Cent Weight of Cleaned Coal	Specific Gravity of Separation	Coal Recovered		Refuse	
			Ash, Per Cent	Sulphur, Per Cent	Ash, Per Cent	Sulphur, Per Cent
Scale Line *OY*	Scale Line *XZ*	Scale Line *GG*	*OX*	*SS*	R_aR_a	R_wR_w
		Graph G_1G_1	C_1C_1	S_1S_1	R_1R_1	S_1S_1
29.2	70.8	1.30	5.5	3.23	42.2	29.2
23.8	76.2	1.35	6.1	3.27	48.6	23.8
20.1	79.9	1.40	6.5	3.30	55.0	19.1
17.3	82.7	1.45	7.0	3.32	60.2	17.3
15.4	84.6	1.50	7.4	3.34	64.5	15.4
12.8	87.2	1.60	8.1	3.35	71.2	12.8
10.5	89.5	1.80	8.9	3.37	78.4	10.5

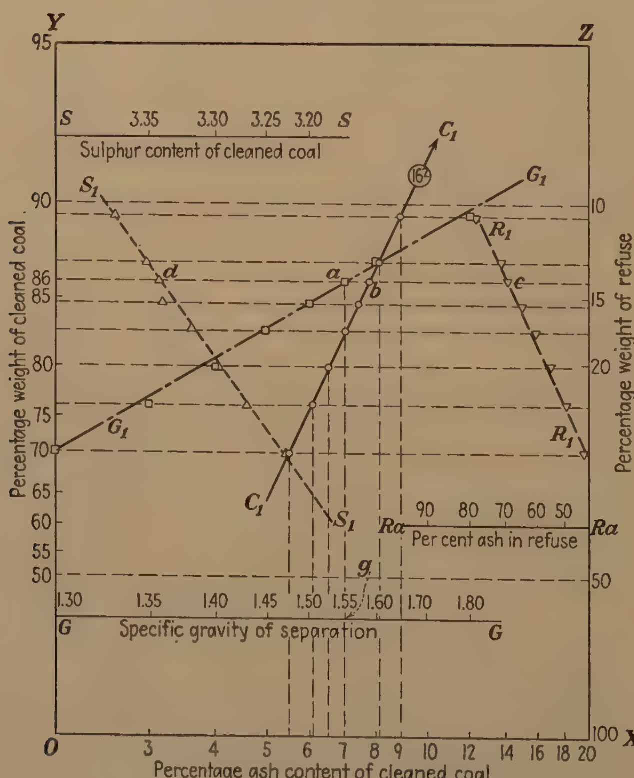

FIG. 2.—WASHING GRAPHS, SINGLE SIZE RANGE.

of 7.7, 68 and 3.34; consequently, a separation at 1.55 sp. gr. would give 85.5 per cent in weight of cleaned coal with 7.7 per cent ash, 3.34 per cent of sulphur and 14.5 per cent of refuse at 68 per cent ash.

Such results are accurate within the gravities of the actual tests; namely, 1.3 and 1.8. Beyond these points, they can be extrapolated more easily than with the old system of curves, but the investigator is necessarily limited to the specific gravities covered by his determinations. In particular, it should be noted that the point at 100 per cent recovery, which is beyond the limits of the graph, is beyond the limits of practical interest.

In this example, the graphs of refuse ash R_1R_1, of sulphur in cleaned coal S_1S_1, and specific gravity G_1G_1 are shown, but usually only the recovery graph C_1C_1 is required, and it only will be shown through the remainder of the paper. It will give all the data required, and the presentations thereby will be clarified.

Characteristics of Graphs of Single Size Ranges

When a graph is plotted on an arithmetic scale rather than on a logarithmic basis, the useful portion of the washability curve is often reduced to a short segment, difficult to characterize and to interpolate. The values of the constants a and n of the power curve are not in evidence. On the other hand, when logarithmic scales are used, the useful portion always can be amplified at will, and the characteristic constants are easily determinable.

To illustrate this argument, Table 2 shows the washability values of 11 coals from widely selected areas.

Float-and-sink characteristics of these 11 coals are represented in Fig. 3 by washability curves plotted according to the old method of presentation and in Fig. 4 by the new logarithmic method. In Fig. 3 the curves overlap and no definite characteristics can be discerned, but in Fig. 4 the graphs of the several curves are shown clearly and distinctly. They group themselves into separate zones, according to their diverse characteristics:

Class A.—Coals 1, 2, 3 and 4 have low inherent ash. 1, 2 and 4 are relatively clean; 3 is dirty (extraneous impurities).

Class B.—Coals 5, 6 and 7 have higher inherent ash. They can be cleaned down to only 6 or 8 per cent ash. Coal 5 has a relatively high percentage of extraneous ash, whereas coal 7 is relatively free of such material.

Class C.—Coals 8, 9, 10 and 11 have high inherent ash percentages and, in the order in which they are listed, have increasing

Table 2.—*Gravity Separation of Various Coals*[a]

No.	Name	1.40 Sp. Gr.		1.50 Sp. Gr.		1.60 Sp. Gr.		Total	
		Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent
1	Northumberland DCB 2 to 1½-in.	87.7	2.4	88.9	2.5	90.3	2.94	100	14.0
2	Northumberland C 2½ to 2-in.	76.3	3.3	78.5	3.7	79.7	4.1	100	17.5
3	Northumberland C ⅛ to 1/16-in.	59.6	3.1	61.5	4.2	62.8	5.05	100	27.6
4	Pittsburgh seam, Pa. 3-in. to 0	87.0	4.03	90.7	4.5	93.0	5.1	100	9.38
5	Yorkshire, England, 3-in. to 0	59.8	6.2	65.2	6.9	67.9	7.4	100	27.30
6	No. 3 bed, Indiana, ¼-in. to 0	79.9	6.5	84.6	7.4	87.2	8.1	100	18.70
7	Miller seam, Pa., 3-in. to 0	90.7	7.2	93.2	7.7	94.7	8.05	100	10.76
8	Mary Lee seam, Alabama, 3-in. to 0	76.5	10.5	82.0	11.9	82.0	12.0	100	22.60
9	Bon Air bed, Tennessee, ¼-in. to 0	80.5	10.6	86.0	11.2	91.5	12.2	100	15.10
10	Raton bed, New Mexico, 1½-in. to 0	64.5	10.9	74.9	12.7	84.9	15.1	100	20.70
11	V. Island ⅛-in. to 0	42.4	11.3	53.7	14.7	61.4	17.4	100	34.30

[a] Data on coals 1, 2 and 3 from The Cleaning of Coal, by Chapman and Mott; coals 4, 5, 7 and 8 from W. Clark, *Bull.* Can. Inst. Min. and Met., 34; 6, 9, 10 and 11 from U.S. Bur. Mines *Bull.* 300.

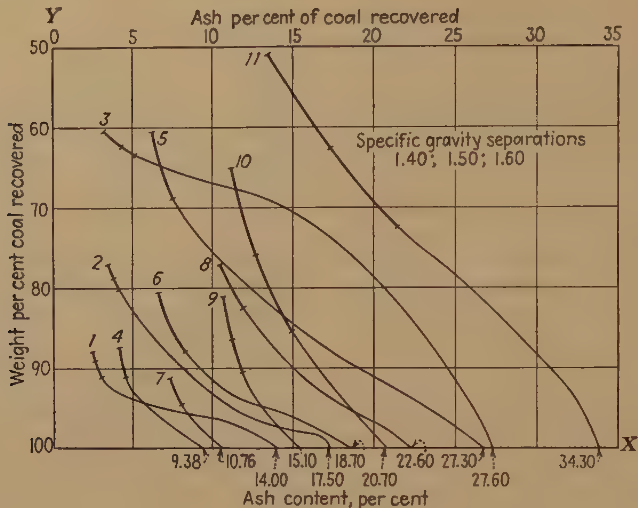

FIG. 3.—WASHABILITY CURVES.

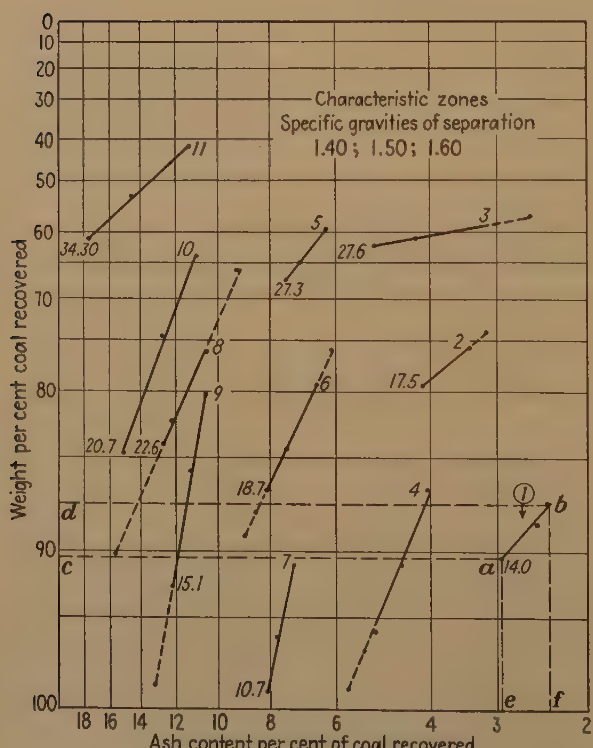

FIG. 4.—ELEMENTARY WASHING GRAPHS.

percentages of impurities at 1.40 specific gravity.

Difficulty and Economy Factors

Other interesting characteristics may readily be discerned. For instance, the relative difficulty of washing a mineral at any specific gravity may be measured to some extent by the quantity of that mineral that has a specific gravity differing, in either direction, from the specific gravity of separation by less than some specific figure. This value will be designated the "factor of difficulty." Thus, if the specific gravity of separation be assumed to be 1.50, the difficulty of washing may be regarded as proportional to the entire quantity of mineral having a specific gravity lying between 1.4 and 1.6. For coal 1, that quantity is 90.3 less 87.7 or 2.6 per cent. It is the vertical projection of the line *ab* in Fig. 4, coal 1, as determined by the intercept *cd* on the vertical axis by horizontal lines drawn from *a* and *b*. From these graphs can be determined also the economy of washing at any specific gravity. This will be designated "the economy factor." It is the percentage of weight of loss, corresponding to an

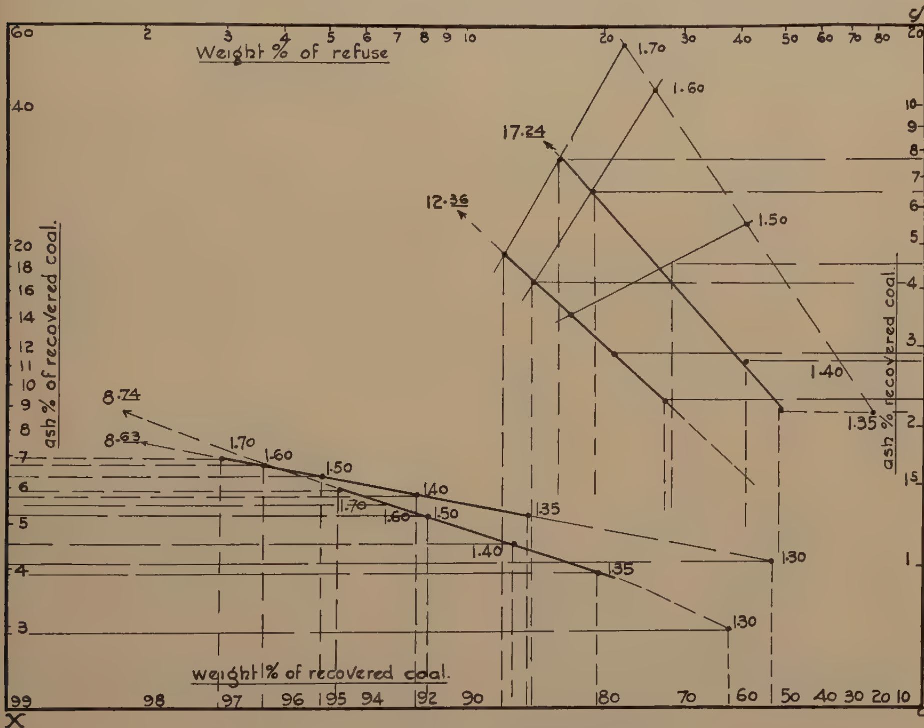

Fig. 5.—Elementary washing graphs.

improvement of 1 per cent in the ash content of the cleaned coal; or, returning to Fig. 4, the ratio of *cd*, vertical projection of the vector *ab* representing the coal, to *ef*, horizontal projection of that vector; or *cd* ÷ *ef*. For coal 1 in Fig. 4, *cd* = 2.6 and *ef* = 0.54, so the ratio, or economy factor, is 2.6 ÷ 0.54 = 4.82.

These factors of difficulty and economy for the 11 coals are recorded in Table 3. Classes A, B and C, as already defined, are arranged to correspond to a gradually increasing inherent ash.

Any difficulty factor lower than 5 indicates a coal that is easy to clean. With such a coal even a crude operation will have

small losses. When the factor is above 10, the coal is cleaned only with real difficulty. One having a factor exceeding 20 requires a very accurate method and careful operation. For further development, the able work of B. M. Bird, of the Battelle Memorial Institute, should be consulted. We have tried only to simplify the measure of the difficulty, by referring to vectors, easier to read on our scales, amplified within the working ranges.

TABLE 3.—*Difficulty and Economy Factors for the Eleven Coals in Table 2*
1.50 SPECIFIC GRAVITY

No.	Class	Difficulty Factor	Economy Factor
1	A	2.6	4.82
2	A	3.4	4.25
3	A	3.2	1.64
4	A	6.0	5.61
5	B	8.1	6.75
6	B	7.3	4.56
7	B	4.0	4.71
8	C	5.5	3.66
9	C	11.0	6.88
10	C	20.4	4.85
11	C	19.0	3.11

In considering the economy factor, each case must be decided on its own merits. To establish a basis of comparison, the cost of coal at the tipple may be assumed to be \$1 per ton, and the increase in sales value for each 1 per cent in ash reduction at 5¢ per ton. In such a case, there will be neither profit nor loss in cleaning (neglecting operational expense) if the economy factor is 5; for, with such factor, to a decrease in ash percentage equal to 1, and worth 5¢, corresponds a loss in weight equal to 5 per cent and worth 5¢. Thus, with the assumptions set forth and within the gravities considered, coals having an economy factor below 5 can be washed with economy. With coals having a higher factor, washing is undesirable, unless the coal would be unmarketable without cleaning.

GRAPHS OF COMPOSITE COALS

Unless the characteristics of coals are determined for a narrow size range, they are likely to have wide fluctuations. Hence, it is necessary to regard and treat the average coal sample as a composite of a number of units, all with their different characteristics. This contention is illustrated by the data in Table 4, in which are grouped together at random four of the sizes of a given coal; namely, coal No. 6 from Franklin County, as given in *Bulletin* No. 217 of the University of Illinois—a fair example, supported by others that follow.

This example shows that the percentage of clean coal at the various specific gravities considered varies, according to sizes, between 52 and 86 per cent at a specific gravity of 1.35; 60 and 92 per cent at 1.40; 72 and 95 per cent at 1.50; 81.4 and 96.4 per cent at 1.60 and 84.6 and 97.1 per cent at 1.70.

The ash contents also at these specific gravities vary for these sizes from 2.3 to 5.1; 2.8 to 5.7, and so on.

Experience at most mines shows a wide variation in the size consist of any unscreened coal. This is due not only to underground conditions but also to unavoidable size segregations that accompany successive loading and unloading opera-

TABLE 4.—*Float-and-sink Data for Coal from No. 6 Bed, Franklin County Illinois*

No.	Size	Ash, Per Cent	1.35 Sp. Gr.		1.40 Sp. Gr.		1.50 Sp. Gr.		1.60 Sp. Gr.		1.70 Sp. Gr.	
			Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent
1	½ to ¾ in.	8.63	86.5	5.1	92.3	5.7	95.2	6.3	96.4	6.7	97.1	6.9
2	3⁄16 in. to 10 mesh	8.74	80.8	3.9	87.3	4.5	91.8	5.2	93.5	5.5	94.8	5.9
3	20 to 48 mesh	12.38	73.0	2.3	79.2	2.9	83.3	3.5	86.2	4.1	88.0	4.75
4	Through 48 mesh	17.24	51.9	2.3	59.6	2.8	71.7	4.6	81.4	6.5	84.6	7.6

tions. If, however, a narrow size range is taken, the coal will show consistent characteristics. Recoveries will not vary more than 1 or 2 per cent, and ash contents not more than 0.5 per cent. But the characteristics of an unscreened coal, being made of a variable proportion of the several size ranges, will be so divergent as to be of no practical value when obtained from float-and-sink tests made of the bulk sample.

The only representation of a composite coal having any practical value and giving consistent, definite characteristics, will be one obtained by bringing together in condensed form all its various elements. Instead of a single characteristic graph, there should be a complete group of all the component graphs.

Typical Composite Coals

All close observers have fully realized the need for segregation of a composite sample into its elementary sizes. Most of the latest washability studies presented by the U.S. Bureau of Mines have been so conducted, the sizes generally used being 3 to 1½ in.; 1½ to ¾ in.; ¾ to ³⁄₁₆ in.; ³⁄₁₆ in. to 14 mesh; 14 to 35 mesh; 35 to 100 mesh and through 100 mesh. This is a practical and consistent scale, with a ratio of sizes within each group not exceeding 2 to 1 in the larger sizes, and not much more in the smaller sizes. This fits the average coal well.

However, with the present system of graphic presentation, these seven sizes plus the composite sample with five curves for each will give a total of 40 washability curves. By condensing these into a few pictures, a much clearer view will be obtained and many new relationships be revealed. These comprehensive pictures will supply a new method of identification and will be of great value in weighing the merits of different coal-cleaning methods and in the interpretation of operating results.

By drawing lines to join points on the graphs that represent the recoveries at the same specific gravity, a solid presentation of the coal sample is obtained. This is a three-dimensions representation: X = ash content of cleaned coal, Y = weight per cent of this coal, and Z = specific gravity of separation. The lines of equal specific gravity resemble the contour lines, or lines of equal elevation used in topographical charts. They are the intersections of the solid representing the particular coal by horizontal planes corresponding to the various specific gravity separations considered.

Characteristics of a Solid Presentation

Our first example is taken from *R.I.* 3083, of the U.S. Bureau of Mines. The six sizes analyzed are represented by the seven lines 1 to 6, corresponding to sizes 1½ to ¾-in., ¾ to ³⁄₁₆-in. . . . down to minus 100-mesh. Ash contents of the raw sizes are shown in a circle at the top of each line: 18.2, 10.9 . . . 17.6. (Fig. 6.)

The solid presentation of the Black Creek coal as a whole is of a wide U-shape, similar to a trough or to a sheet of paper partly doubled on itself. The left half is almost vertical and the right half spreads to the bottom of the chart. This folding may be termed the "rotation" of the figure, which may be defined as the direction followed by the specific gravity ("contour") lines when going from the largest size to the smallest; in this case from 1 to 6. This is an important characteristic feature of the figures. The specific gravity curves in this instance have a clockwise rotation.

A seventh recovery line has been plotted representing the recoveries of the composite sample. Each point of this composite line is like the center of gravity of the corresponding specific gravity line, but the real value of this line is only to show at a glance how the various sizes actually are distributed in this particular sample. In this instance the location of the composite line shows clearly the importance of the large sizes; the small

sizes form only a small part of the sample represented.

A close examination of Fig. 6 suggests the following conclusions: All sizes are able. The small sizes, from 3 to 6, should be cleaned to give 3.5 per cent ash. The specific gravities that will give such a separation can be determined by noting

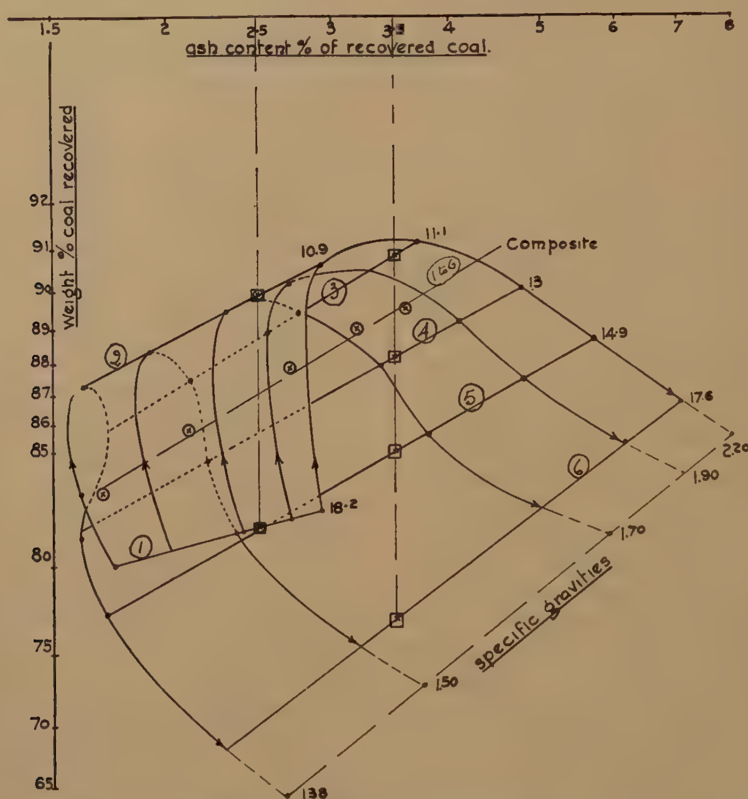

FIG. 6.—COMPOUND WASHING GRAPH.

From Washability Studies of Black Creek Bed at Bradford Mine, Dixiana, Alabama. U.S. Bureau of Mines *Report of Investigations* 3083.

rather dirty, but the inherent ash is low, mostly in the largest sizes. All "coal unit" lines are rather steep, indicating coals easy and economical to clean. A logical solution for the cleaning of this coal might be as follows: The large sizes 1 and 2 (1½ to ³⁄₁₆ in.) which constitute more than 50 per cent by weight of the entire coal, should be cleaned in equipment capable of making an average separation at 1.70 with satisfactory efficiency. This is an easy washing zone for these coals, and a product with 2.5 per cent ash should be readily obtainable.

where a vertical line passing through the point representing 3.5 per cent ash will intersect the several washability graphs. The points thus determined have been surrounded by squares. The relation of these points to the specific gravity lines will make it possible to interpolate the several specific gravity values. Thus size 3 should be cleaned at a specific gravity of 2.05, size 4 at a specific gravity of 1.72, size 5 at 1.65 and size 6 at 1.52.

This example illustrates clearly the fallacy of a separation of all sizes at a single

specific gravity. If, for instance, all these four sizes were cleaned at a specific gravity of 1.90, the ash contents of the resultant products would vary from 2.55 to 6.25 per cent; that is, in the proportion of 1 to 3—obviously a very undesirable preparation. Yet coals that would be represented by a figure similar to this one are frequently found in Canada and in the United States. In fact, the proportion in weight of the smallest sizes is generally much larger than this, with practical consequences that would be more serious.

In a single jig, the gradually decreasing sizes cannot be separated, as required here, at a gradually decreasing specific gravity. Equipment operating on entirely different principles is required—a wet table, for instance.

Various Shapes and Characteristics in Solid Presentation

Fig. 7 (based on U.S. Bureau of Mines *R.I.* 3450) deals with the same bed, but at a different mine. It illustrates the claim that the general shape of the figure is an outstanding characteristic of the coal bed. However, at Yolande No. 6 the ash characteristics are different from those at the Bradford mine; the inherent ash content is higher, and the impurities as a whole lower, but the shape is still that of a folded sheet, and the rotation is still clockwise. However, here the left branch is more extended than the right, partly because one more large size, 3 to 1½-in., is included.

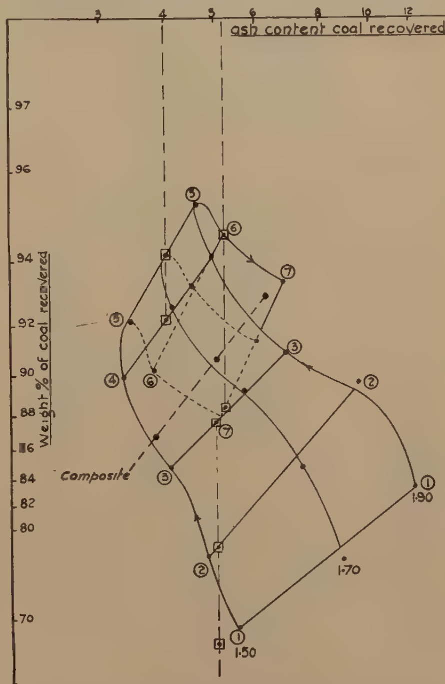

Fig. 7.—Compound washing graph.
Black Creek bed, Yolande No. 6, Rockcastle, Alabama. U.S. Bur. Mines. *R.I.* 3450.

The location of the line representing the composite coal suggests a greater importance of the small sizes. The fold occurs here at a much smaller size, 14 to 35-mesh,

Table 5.—*Coal from Black Creek Bed, Bradford Mine, Dixiana, Alabama*[a]

No.	Sizes	Weight Per Cent M.R.	Float at 1.38 Sp. Gr.		Float at 1.50 Sp. Gr.		Float at 1.70 Sp. Gr.		Float at 1.90 Sp. Gr.		Float at 2.20 Sp. Gr.		Total	
			Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent	Wt., Per Cent	Ash, Per Cent
	Inch													
1	1½–¾	18.2	78.9	1.8	80.7	2.0	81.7	2.4	82.2	2.7	82.5	2.9	100	18.2
2	¾–3/16	42.1	85.9	1.6	88.3	1.9	89.5	2.3	90.2	2.7	90.6	2.9	100	10.9
3	3/16 to 14 mesh	30.0	84.4	1.7	87.4	2.1	89.3	2.7	90.5	3.3	91.2	3.7	100	11.1
	Mesh													
4	14 to 35	8.1	81.3	1.6	84.7	2.2	87.9	3.4	89.2	4.1	90.1	4.8	100	13.0
5	35 to 100	4.4	77.5	1.7	81.8	2.4	85.6	3.8	87.5	4.8	88.7	5.7	100	14.9
6	Through 100	2.6	68.1	2.3	75.5	3.2	83.2	5.0	85.3	6.2	86.7	7.1	100	17.6
1 to 6	Composite	100	83^3	1^7	86^1	2^1	87^9	2^7	88^9	3^2	89^5	3^6	100	12^5

[a] Figures from U.S. Bur. Mines *R.I.* 3083.

whereas in the diagram for the coal from the Bradford mine the fold occurs along the washability graph that applies to the coal of size between ¾-in. and 14-mesh.

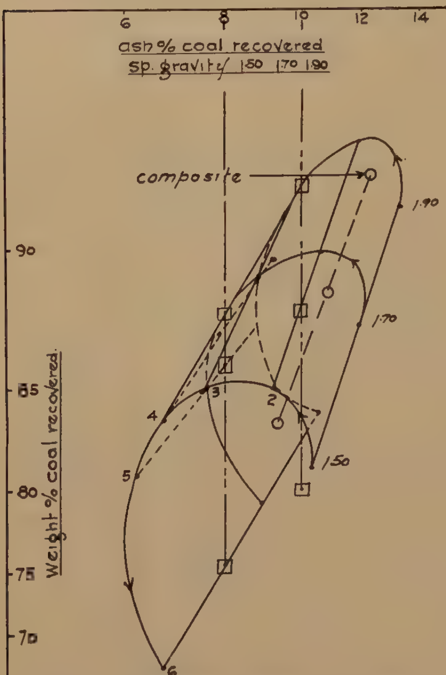

FIG. 8.—COMPOUND WASHING GRAPHS.
Brookwood bed. Warrior View mine, Tuscaloosa, Alabama. U.S. Bur. Mines *R.I.* 3170.

But the same practical washing conclusions are valid, and they agree entirely with the conclusions on page 7 of Bureau of Mines *Report of Investigations* No. 3450: "Coal-washing practice in Alabama indicates that somewhat more efficient cleaning can be obtained on 3-in. to 0 coal, by screening out the fines and treating them separately." The authors of the report conclude by suggesting that the sizes above 3⁄16-in. be jigged and that below 3⁄16-in. be tabled, which is exactly in line with the conclusions suggested by a study of the solid presentation diagram.

Solely on the basis of technical and economic factors, the diagram in Fig. 7 suggests the following washing procedures: Sizes 1, 2 and 3 (3 to 3⁄16-in.), constituting about 42 per cent of the entire weight of the coal, to be washed in a single jig at 1.47, 1.51 and 1.62 sp. gr., respectively. Such a

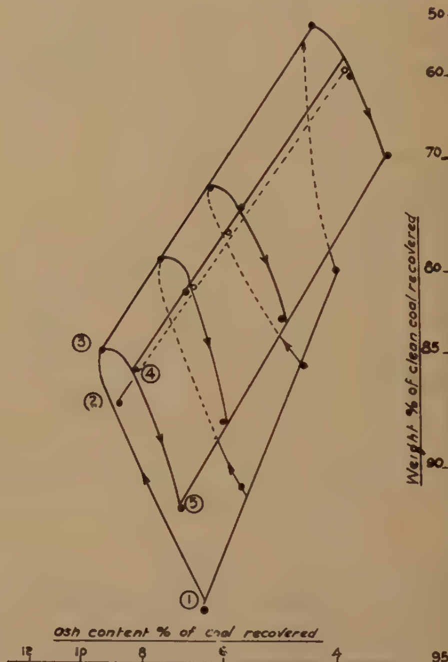

FIG. 9.—COMPOUND WASHING GRAPHS.
Western Canada, Coal M. American Coal Cleaning Corporation *Test Report* No. 147-A.

graduated range can easily be obtained in any modern jig, thus affording in this instance, as can be seen by the ordinate dropped from 5, on the line for ash content of coal recovered, a uniform ash of 5 per cent on all three sizes.

Sizes 4 and 5 (3⁄16-in. to 35-mesh), or 47 per cent of the entire coal, should be washed in a separate box at 1.67 and 1.70 sp. gr., respectively, to segregate coal of 4 per cent ash, and this again is an easy operation in the average modern jig. Sizes 6 and 7, which include all coal under 35-mesh, should be washed at 1.90 and 1.50 sp. gr. separations, respectively, to obtain a coal of 5.20 per cent ash, an operation that will require equipment working on a different principle from that of the

ordinary jig, because decreased sizes cannot be cleaned in a jig at decreased gravity. It may be pointed out that coal at 35-mesh is almost always screened wet. Sizes 6 and 7 may be passed through the same jig as the sizes 4 to 5, but will come out virtually uncleaned. However, they can be separated on the dewatering screen of jig No. 2 and rewashed in suitable equipment.

The third example, Fig. 8, is taken from the Brookwood bed, at the Warrior View mine, as analyzed in U.S. Bureau of Mines *R.I.* 3170. The specific-gravity lines have a counter-clockwise rotation. This coal, though it has a high inherent ash, is relatively clean, but is difficult to wash. Because of its rather narrow spread, it can be cleaned only by subjecting it to treatment in two pieces of equipment. The large sizes, 1, 2, and 3 (1¼-in. to 14-mesh), can be washed easily in a single jig at the respective gravities 1.48, 1.62 and 1.92, to deliver a uniform 10 per cent ash coal. The rest of the sizes should be washed at 1.75, 1.72 and 1.60 sp. gr., respectively, to obtain a uniform clean coal at 8 per cent ash.

Fig. 9 illustrates a coal from western Canada with an exceptional clockwise fold or rotation, and a narrow "dispersion." On the basis of an identical number of standard elementary sizes, the area covered by the diagram is an important characteristic of the coal represented. A wide dispersion, as in Fig. 7, indicates the necessity of screening first and then washing in several apparatus; with a narrow dispersion, as in Fig. 9, "washing first" all sizes in one single apparatus is more likely.

Unrepresentative Sample

Truly representative samples, undisturbed by crushing, and carefully tested for float and sink on a large scale, have always given a definite characteristic solid diagram. The shape of this solid and the direction of rotation of its lines of specific gravity, from larger sizes to smaller, are among the most consistent characteristics for the identification of the particular coal seam, giving at a glance the most complete, helpful and accurate solution for the cleaning of the coal represented. If the several plotted points will not join without discrepancy, further investigation usually will show some irregularity, or a gross mistake has been made in either sampling or testing.

Acknowledgment

The author wishes to tender his thanks to Dr. H. F. Yancey and Mr. M. R. Geer, of the U. S. Bureau of Mines at Seattle, for their encouragement and valuable assistance in the preparation of this paper.

Ventilation at Mines of the Lehigh Navigation Coal Company Inc.

By A. T. Beckwith,* Member A.I.M.E.

(New York Meeting, February 1942)

The Lehigh Navigation Coal Company Inc. operates steep-pitch, relatively deep mines in the Panther Creek Valley, at the eastern end of the southern anthracite coal field. Commercially minable coal beds are the Mammoth,† Primrose, Orchard, and Buck Mountain;‡ their respective thicknesses average 50, 15, 10 and 12 ft. Sand slates immediately overlie and underlie these beds, though in some locations sandstone or conglomerate beds occur. Underground operations extend to a depth of 1200 ft. below the surface, and comprise a bulk of virgin work on the lower levels, and a small amount of remining on the upper levels. With increase in depth, ventilation has become a more important factor.

Development

In early days slopes or drifts were driven in the outcropping coal on the hillsides. Subsequent development was by water-level tunnels, with portals above high water in Panther Creek. These tunnels have been maintained because they intercept and drain much water that enters the mined-out portion of the bed and otherwise would follow down to the deep workings. Below these tunnels, the coal was customarily developed by slopes. Muleways, which are zigzag chutes easy to travel, were also driven to connect lower openings with the water levels.

When it became necessary to go still deeper, the slopes were abandoned, and vertical multicompartment shafts were sunk from the surface. Levels have been at vertical intervals of from 175 to 250 ft. Recent practice has been toward the latter spacing, splitting the total lift, however, by starting to mine from chutes extending halfway up to the level above. Chutes customarily are spaced at 50, 60 or 70-ft. centers along the gangways.

At some mines, additional shafts have been sunk from the surface to the level containing the main pumping station, and these carry the discharge lines from the pumps. As the mines are ventilated by the exhausting system, both the hoisting and so-called water shafts serve as intakes.

The main fan is always on the surface, and is connected with the return airways from the various levels by an air hole or a vertical shaft. For economy, when the present deepest levels were developed, main return-air shafts or air holes were not extended to the new level, but connections were made from the new level to that above by two or more holes. It is doubtful that this method will be continued, as complications in the system, increased resistance to flow, and other difficulties undoubtedly will counterbalance any economies this provision may afford.

During the early years of mining, when operations were at no great depth, all development headings were driven in the coal bed. As mining progressed downward, increased ground pressures slowed develop-

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† About 80 per cent of the mine production now comes from the Mammoth bed. Although there is a marked similarity in the method of working the other beds, the discussion in this paper should be considered as referring to the Mammoth, to avoid confusion in interpretation.

‡ The Buck Mountain bed is minable only at the ends of the Panther Creek Valley.

ment, and maintenance costs became excessive. As a result, gangways and airways were driven in the Skidmore bed, a leader underneath the Mammoth, and rock chutes were driven from the gangway to reach that bed for mining. To provide split ventilation, overcasts were introduced at about every fourth chute. As under such an arrangement breast pillars could be recovered on the advance, starting from the top down, this became general practice.

In recent years, severe ground pressures in many localities have made upkeep costs of Skidmore gangways and airways unduly high. Consequently, in developing some of the newest levels, both headings have been placed in a bed of solid rock below the Skidmore, where such openings will stand with little or no timber. Return airways are at the same elevation as gangways, and have the same cross-sectional area. This layout has many advantages; i.e., the air gangways can be driven by the standard methods for such work, inspection is easier, cross-sectional area can be made larger than heretofore, and, inasmuch as track installed while driving is not lifted, any falls that occur can be loaded into cars that reach the air gangway through a connecting tunnel off the haulageway.

Mining

The basic method of mining is virtually the same now as in early years, but modifications are made constantly to cope with changing bed conditions. Two miners of equal standing work together on a contract basis, being paid for yardage driven or cars loaded. Fig. 1 shows a system employed in mining the Mammoth bed from one of the deep levels, and will serve to explain the method of approach.

A breast is first opened by cutting out a slice of coal 18 or 24 ft. wide, 15 ft. high, and extending from the bottom to the top rock of the bed. Manways are provided along each side of the breast, and serve as both traveling ways and air courses. The miners stand on top of the broken coal to advance the face, and thus the system is essentially the same as shrinkage stoping. A succession of cuts up the pitch and toward the back advances the breast, which finally breaks through into the gob rock. The broken coal in the breast is then drawn out.

Pillars left between the mined-out breasts are recovered by roughly traversing the pillar with a system of chutes, so arranged that the coal can be extracted as a series of separate blocks. Starting at the top, one block of the pillar at a time is recovered by opening a small hole or pillar breast at some convenient point along a coal chute, and constantly enlarging this opening, until only a heavy shell of coal remains between the opening and the gob. The shell is drilled and blasted ("tumbled"). Then, as in the drawing of a completed breast, the broken coal is run out of the battery until rock appears, whereupon this "tap" is finished and a new one is started.

Frequently while a breast is being opened or driven, the coal will cave, and may continue to run through to the gob above. Such running may be confined to a soft bench, or may take any irregular path. In any event, much coal may be left in place, and because of the difficulties involved in mining such coal during pillar work, a considerable amount may be left unrecovered when the miners abandon their chute. The fact that broken rock overlying broken coal will quickly work its way down through the coal, and leave some of the broken coal trapped above, is an added difficulty in obtaining a satisfactory recovery.

Twenty years or so after the coal tributary to any level has been extracted, the gob will have ceased to shift and be tightly packed. If remining is justified, it is done by driving chutes through the mass of tightly packed fine coal dirt, rock, old timber, and solid coal. When patches of solid coal, or pockets of the broken material are en-

countered, mining proceeds in a manner similar to that used in recovering virgin breast pillars.

Conditions Requiring Ventilation

In deep mining, methane is constantly emitted from all fresh coal openings, and gas blowers often cause large local concentrations. Thus the main ventilation problem is to keep the working places clear from firedamp so that miners can work and work in safety. Occurrences of black damp are not frequent; they are largely confined to the areas in the upper levels that are being remined and have only natural ventilation. In general, no other gases are found. Each place is inspected daily by a fire boss before miners start work, and in addition, each party of miners constantly keeps a safety lamp burning at the working face to detect accumulations of methane.

Operations such as actual mining, running coal down chutes, and loading from chute spouts into mine cars, may create and disperse fine dusts, especially in virgin work on deep levels, where, unless the ventilating current is sufficiently strong to sweep the fine black dust away quickly, visibility with the customary permissible electric head lamps may be limited. Miners, battery starters, or loaders must then wear respirators, and cease work until the dust clears.

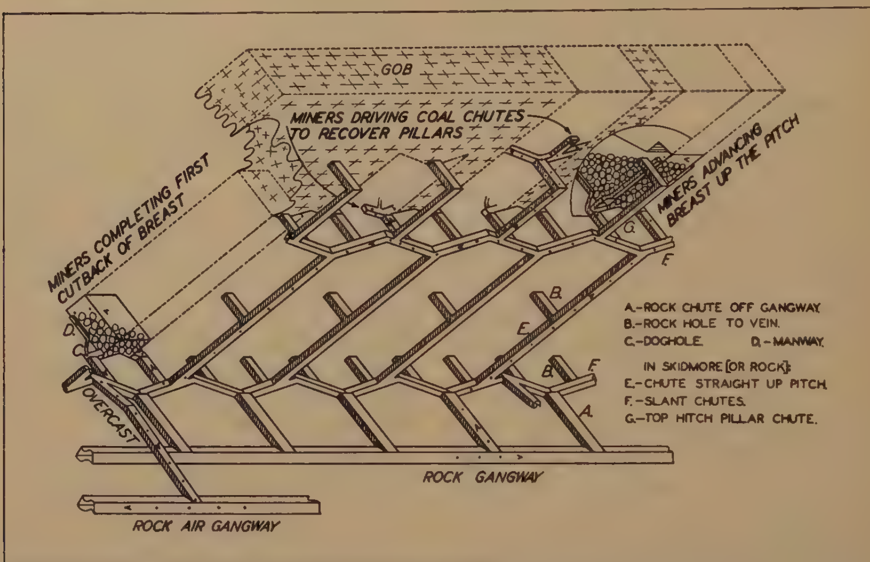

Fig. 1.—Isometric view of one method of mining. Ventilating current is shown by dotted lines.

In an effort to combat dust created during loading, experiments have been made with water sprays. Although they helped to lay the dust, the loaders objected strongly to the spray blown on them by the ventilating current. Undoubtedly the problem will receive further attention; it is limited to certain kinds of advance work, because on steep-pitch mining, the material is usually damp.

Mining work is planned so that most of the coal can be blasted at the end of the shift. As there is but one mining shift, with a consequent lapse of several hours between the time of blasting and the resumption of work at the face, the removal of

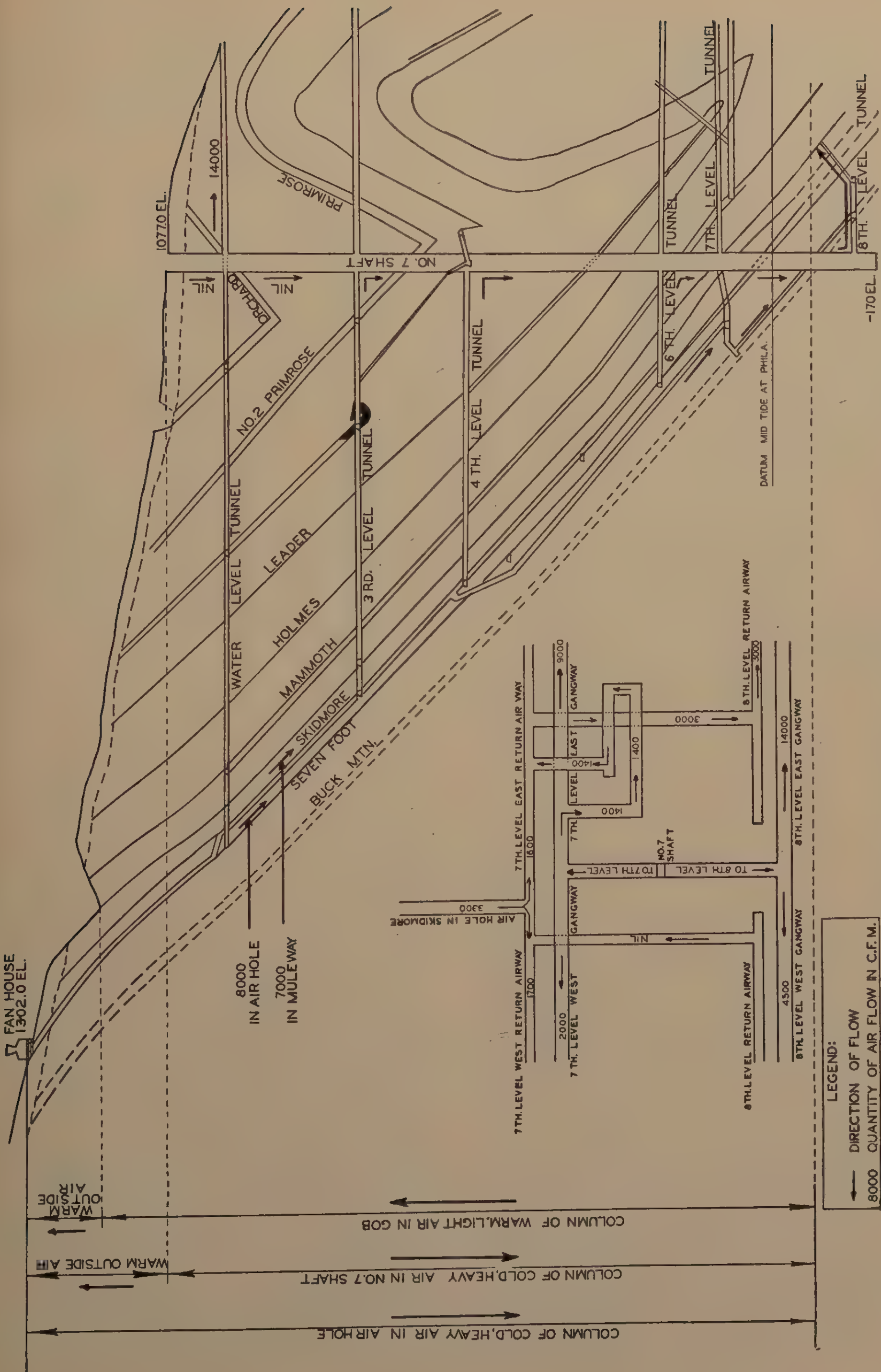

FIG. 2.—VERTICAL SECTION WITH INSET HORIZONTAL SKETCH, SHOWING NATURAL FLOW AT 86°F. OUTSIDE.

blasting fumes presents no great problem. Removal of powder smoke from rock-development headings will be discussed under auxiliary ventilation.

In worked-out areas, where coal has been left in place and the surrounding gob is packed, heat generated by slow oxidation of the coal, timber, or other causes, is not easily dissipated. This gob heat may be encountered in remining, or in virgin work upon breaking an opening into such gob. The atmosphere is usually devitalizing, saturated, and temperatures up to 125°F. have been recorded. Such conditions have been handled thus far by providing as large a split of air as possible, and by cooling the working either by a jet of compressed air or by carrying auxiliary ventilation to the face through a conduit.

Ventilation also affects the economical operation of the mines by its effect on timber. Yet, in spite of the staggering costs for replacements of such mine timber as has failed through dry rot, little definite information is available on how air composition, velocity, temperature, and relative humidity affect timber installations.

Primary Ventilation

As in other sections of the anthracite field, early ventilation in the Panther Creek Valley was by natural flow, supplemented by furnaces and hand fans, which later were replaced by centrifugal fans. Now the propeller fan has found a place in the scheme, and probably will be used for all future installations. In most of the remining work in the upper levels the air is now circulated solely by natural ventilation.

A general understanding of the primary system may be gained from Figs. 2, 3 and 4. These show No. 8 mine at Coaldale, where coal is being remined on the Third and Fourth levels, and where the Seventh and Eighth-level workings are in virgin coal. The mountain fan (an exhaust unit) ventilates only the Seventh and Eighth levels. Because of the many openings in the upper portions of the mine, the natural coursing of the air may be complicated. With proper attention, however, all requirements can be met.

Fig. 2 shows circulation of air with mine idle and main fan stopped, and when outside temperature averaged 86°F. Air on Seventh and Eighth levels was 60° ±, and temperatures in portions of the air hole, shaft, and other places very near the surface, were nearer 60° than 86°. The columns of air in the No. 7 shaft and fan hole, therefore, were relatively cold and heavy, whereas the air column in the gob was relatively warm, and consequently light. (It will be remembered that the mined-out portion of the bed extending from the deepest levels to the surface is filled with gob, which may generate much heat.) Because of differences in the weights of the interconnected air columns, a natural flow is established, having the shaft and air hole as intakes and the gob as an upcast.

Although 15,000 cu. ft. per min. is entering the fan hole at the surface, the door separating the top of the muleway from the fan hole is not kept as tight as when the fan is running, for the customary fan pressure is not present to keep the door firmly closed. Hence, 7000 cu. ft. per min. is leaking into the muleway and finding an outlet on the Fourth or Sixth level. A similar condition between the fan hole and Fourth-level gangway accounts for 8000 cu. ft. per min. in the air hole at the water level, and only 3000 cu. ft. per min. at the bottom. Air that flows down the shaft and onto the gangways finds its way into the gob by passing up chutes off these headings, whereas that which flows down the air hole and onto the return air courses works its way into the gob by flowing back through the overcasts and then into the workings.

Fig. 3 shows the primary system with the fan in operation, and under a condition

where the outside temperature is practically the same as that during the survey with the fan stopped. The exhaust fan now draws in air both by the water-level tunnel (W-L.T.), and down the No. 7 shaft. When it reaches the Seventh and Eighth-level bottoms it flows in the gangways, is coursed through the workings by splits, and returns by the overcasts, through the return airways, and up the fan hole. Air volumes are under control, and ventilation is supplied where and as needed. Measurements on the two levels show a total intake of 107,000 cu. ft. per min. and a total return of 80,000 cu. ft. per min. Thus the gob draft is exhausting 27,000 cu. ft. per minute.

In contrast with Fig. 3, Fig. 4 shows the effect on the flow of a change in temperature. Here, with an average outside temperature of 33°F., measurements near the shaft on the Seventh and Eighth-level intakes showed a total quantity of 126,000 cu. ft. per min. at 52°; the sum of return measurements made near the fan air hole was 83,000 cu. ft. per min. at an average temperature of 60°. Thus at 33° outside temperature, 43,000 cu. ft. per min. exhausted through the gob, while at 83° there was but 27,000 cu. ft. per min. Also, the quantity handled by the fan was greater at the lower outside temperature.

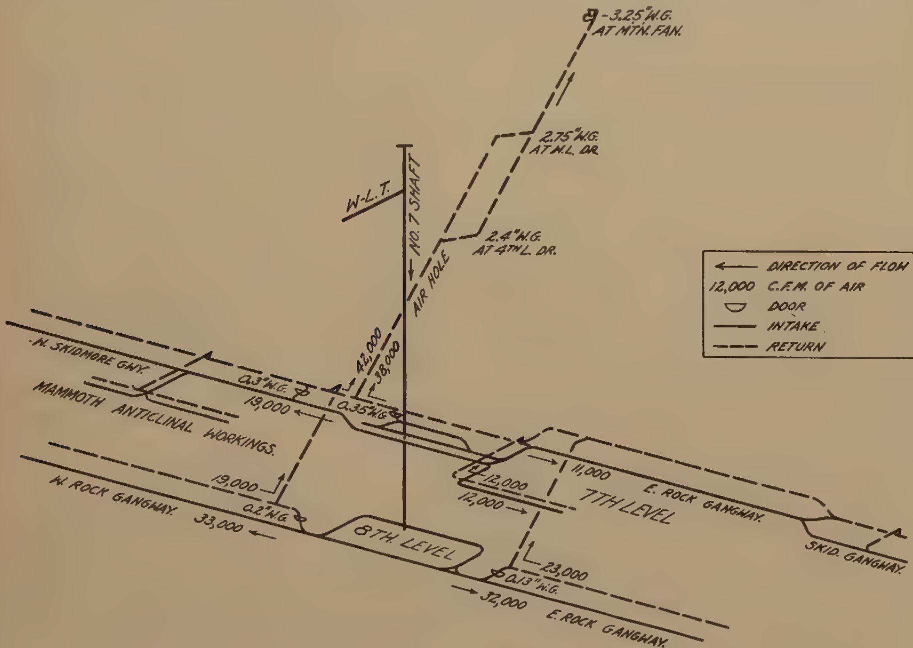

FIG. 3.—METHOD OF PRIMARY VENTILATION WITH FAN IN OPERATION, AT 83°F. OUTSIDE.

At 33° outside and with the fan stopped, natural draft intakes were the No. 7 shaft and W-L.T., and the air returned by air hole and gob.

The No. 7 shaft and other intakes have such a relatively small resistance compared with the fan hole that for practical purposes U-tube water-gauge readings taken at various points along the fan hole have been considered as indicating the drop in pressure along the fan hole itself, rather than for the fan hole plus the intakes. The fan was producing −3.3 in., but the highest differential for any major split was less than 0.5 in. water gauge (Fig. 4). This condition, where so much of the total vacuum produced by the fan was absorbed in moving

the current in the air hole, was known some years ago, and plans were made for a revision of the system. Other considerations, coupled with the major desire to supply a greater quantity more efficiently, have resulted in: (1) a new air hole from the Seventh level to the surface, (2) a new type exhaust fan at the top of this hole, and (3) a revised system, wherein all east-side mine ventilation is handled by the old hole and fan, and the west side by the new. This system is in operation but as yet is being used under a temporary arrangement, as work in certain sections of the hole, such as enlarging, placing steel timber, and eliminating by-passes, is not entirely completed.

An interesting incident accompanied the change over from the old system, when water gauges, and hence quantities, were increased on each major split. With the higher water gauges, east-side measurements showed less air on the intakes than on the returns. Apparently, the increased effective pull by the fan had overbalanced the gob updraft, with the result that air was drawn down through the gob and into the mine, whereas before the gob had served as a power-free exhaust fan. The change was made in the summer, at which time gob draft is at a minimum. The effect of cold weather with the changed conditions has not yet been determined. However, last winter with the old set-up, there was such a strong gob updraft in the Seventh-level E. that the return airway from this section was tightly bratticed, and gob draft used exclusively.

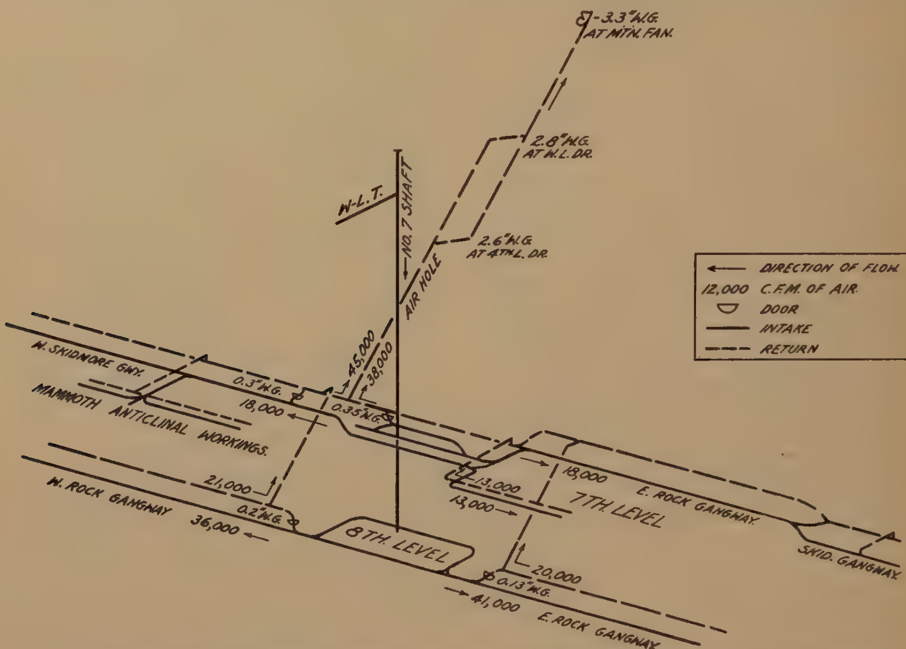

FIG. 4.—METHOD OF PRIMARY VENTILATION WITH FAN IN OPERATION, AT 33°F. OUTSIDE.

As gob updrafts carry away much methane, percentages of this gas in return airways are low, seldom if ever exceeding 0.1 per cent. In view of this condition, return air samples are seldom taken.

In addition to the flow produced by natural ventilation, fans move from 300 to 500 cu. ft. per min. for each employee underground. (Pennsylvania State Mining Laws require that each man underground shall receive a minimum of 200 cu. ft. per

min.). For ventilating the company's five mines in the Panther Creek Valley, nine fans are used; their individual capacities range from 30,000 cu. ft. per min. at minus 1-in. water gauge to 210,000 cu. ft. per min. at minus 4.4-in. water gauge. Combined, they circulate 1,000,000 cu. ft. per min. and use 890 hp.; for the year 1940 they consumed 16.4 per cent of all electric power used for mine service.

Changing market conditions and seasonal demands are reflected by intermittent daily operation of the mines. With broken working time, but continuous fan operation, the amount of air circulated per ton mined is exceptionally high; during recent years the average has been 10 tons of air for each ton of coal produced.

Air Distribution

The general plan for distributing the air current from any gangway to the working faces is shown by Fig. 1. As already stated, overcasts are customarily provided at every fourth place. A split of air leaves the gangway through one chute; the three remaining chutes on the split are blocked at the gangway by a wood trapdoor in the manway side, and by a check brattice, heap of broken coal, and probably a piece of brattice cloth on the chute side. The split is conducted up the chute, through the four working places, then to the overcast, and out by the return air gangway. Return air is never coursed back to the gangway. The brattices used in the slant chute connections to prevent the air from short-circuiting are usually made from 1-in. boards, and contain a slide door of box-regulator type, to permit traveling.

Permanent stoppings are made with cinder-concrete blocks, and have one side faced with a coat of mortar. When miners abandon their chute, the spout is torn off, and such a stopping is installed just above the gangway. Abandoned overcasts, airways, gangways, and other such places are likewise sealed. Where a water trap is provided to carry drainage past a stopping, it is made by omitting one block in the bottom row and building a slightly higher well around the opening on the downstream side of the stopping. Concrete-block frames are used for regulator doors and the doors in panel tunnels. Both doors are single-ply 1-in. ship lath, and the latter are reinforced with strong timber framework and cross braces.

As miners advance their chute or breast, they install the necessary brattices and stoppings to conduct the split of air to their working face. Crosscuts are required upon every advance of 60 ft. in the coal. Chutes have two compartments—the manway for traveling and the chute side, down which the coal runs. Timber sets, with additional props on the line of partition (or line props only in rock chutes) are placed on 5-ft. centers; the line brattice or partition is formed by nailing 1-in. plank to these props. Thus in driving a chute the air is carried up the manway, sweeps the face, and returns down the chute compartment. When rock holes, etc., are abandoned up the pitch, and work is started at a lower point off the main chute, the upper section is sealed off with a plank stopping.

As most of the methane issues from freshly opened coal faces, and the face is always at the highest elevation in any specific opening (tunnels, gangways, and airways excepted), this lighter gas will accumulate there, unless constant ventilation is provided. If a large volume of air were always available, all brattices and stoppings tight, and partitions close to the working face, there would certainly be little trouble in conducting a ventilating current. In the every-day operation of the mines, however, ventilation is interfered with in many ways. The following difficulties are representative but the list is by no means complete:

1. Brattices, traps and sheets sometimes are blown open, jarred loose, or

entirely destroyed by the concussions from blasts made during rock development.

2. During rock or leader-seam chute development, blasting is so heavy that partitions must be kept back a considerable distance from the face.

3. In building chute partitions, especially along the top, planks can be framed only with difficulty around the collars of timber sets, and against jagged coal. Wood stoppings in chutes also can be built tight only with difficulty.

4. Subjected to a squeeze, stoppings may develop cracks, and doors may become jammed.

5. Brattice cloth cannot ordinarily be hung so as to fit tightly.

6. When coal is loaded from a chute, if the chute is entirely or nearly emptied a space is left below the check, which permits air to leak from the gangway.

7. Dogholes, or similar openings have too small a cross section for the proper installation of a line brattice.

8. Feeders may liberate a large volume of methane when coal is being drilled, or after blasting.

9. In breast mining, a fall or run may block the manways.

10. Should one of the first chutes or breasts on a split be holed into the gob, much of the air in the split may be lost in a strong gob updraft; consequently, the places at the other end of the split may suffer.

Auxiliary Ventilation

Small fans, run by 5-hp, or 10-hp. electric motors, were used extensively in past years to blow air through conduits when ventilating certain advancing chutes, tunnels, or other places. Compressed-air injectors are now generally used instead of fans where auxiliary ventilation is required. These injectors, which are either blowing or exhaust units, consist merely of a jet of compressed air, which discharges along the axis of the conduit. The conduit is made of 10-ft., No. 16 gauge galvanized iron pipe sections, having stovepipe ends. The pipes are of 6-in., 8-in. and 10-in. diameter and there are larger sizes for special work. Small-diameter compressed-air jets alone may be used in driving short chutes, or similar places.

Although locally plain conduits containing a jet have been in greatest favor, homemade single-cone, double-cone, and fairly well designed Venturi injectors have been made and used, apparently in the belief that such units would produce exceptional results. To get a comparison of the quantity issuing at the outlet of a conduit, as produced by a compressed-air jet inserted in a Venturi at the intake, against the same jet inserted in the same conduit without the Venturi, tests were run using 1/16, 1/8 and 3/16-in. jets, working under gauge pressures of from 20 to 120 lb. per sq. in., and with 8-in. galvanized conduits, 10 to 100 ft. long. Quantities were calculated from anemometer readings by applying the instrument correction, and by using a factor of 0.85[1] to calculate average velocity, in view of the fact that at the discharge end the anemometer was mounted on the axis of the conduit. The test results showed: (1) The total quantity discharged with the Venturi was no greater than from the plain conduit. (From sundry published articles, the impression is gained that under certain conditions, where large jets are used, the Venturi is useful, but under conditions in these tests no advantage was shown.) (2) The total quantity of air discharged remained the same when the jet was inserted at any location between 3 in. and 16 in. from the back of the conduit with plain inlet. As these test limits were selected at random, it is probable that they do not indicate the full range on the conduit axis, wherein jet location is unimportant.

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[1] G. E. McElroy: Engineering Factors in the Ventilation of Metal Mines. U. S. Bur. Mines *Bull.* 385, (1935) 26.

Tests were made later with plain 8-in. galvanized conduits up to 200 ft. long, to determine the total quantity of air discharged with various gauge pressures, jet sizes and conduit lengths. The results are briefed in the chart of Fig. 6. These tests were made at an elevation of 1000 ft. above sea level, and under the following average conditions: barometer 28.25 in., outside temperature 32°F., and of compressed air 50°. Both tests were made under and for operating conditions, hence refinements (adjustments for these conditions) were not made. Each jet used in these tests was a ¾-in. long brass cone, through the axis of which had been drilled a plain hole of the desired diameter. Jets were made by building up ferrules. The discharge from jets has been taken from "Compressed Air Data."[2]

When making any installation of electrical equipment, care must be taken that it is definitely in the intake air. An added precaution, which cannot be overemphasized, is to give consideration to air recirculation with all types of auxiliary ventilation. This is particularly true with the injector type because, on account of its flexibility and a feeling that it is safe, there may be a tendency to make an improper location. (Locally, electrically operated fans have always been installed on the gangway, while with an injector the conduit can start at the nearest course on fresh air—probably a slant chute well over 100 ft. above the gangway.) To ensure that all areas affected by such units are clear of explosive mixtures, constant inspections must be made by both supervisory force and workers.

In rock-development work, where heavy explosive charges are used, blasting fumes are strong. Most drilling is on the second shift, and mucking, 10 hr. later, on the day shift. Upon blasting, an air jet, placed as close as possible to the face, is turned on, and helps in removing blasting fumes. Only by providing that the last or inbye connection will be as near as possible to the advancing face, can ideal conditions for ventilating such openings be attained.

In the Panther Creek Valley mines, most coal drilling is done with manually operated drills. Compressor capacity for mine service is relatively small, and rock drilling is necessarily staggered, most of it being done on the second shift. Daytime compressed air pressures are customarily low, because of the demands for ventilation, some coal drilling, and miscellaneous uses in the breaker and outside.

Fig. 5.—Equipment used in injector tests.

If great care is not exercised in conducting ventilation to the working faces in mines where compressed air is available, the miners will open the ends of their air lines and use large jets. One or two lines, or several large jets thus opened, may so reduce the compressed air pressure that miners who may customarily get satisfactory service with small jets will find it necessary to use large ones or open-end lines to get more air. Thus the bad practice will spread, picking up momentum, until the point is reached where the pressure is at its lowest possible level.

[2] F. W. O'Neil: Compressed Air Data, 4th Ed., 66, 1934.

If compressed air is to be used at all in connection with auxiliary ventilation, only the jet system should be used, and preferably within a conduit. An illustration of the improvements that can result from an intensive campaign to better ventilation and reduce the waste of compressed air, is shown by the charts in Fig. 7. These are representative compressed-air pressure charts, before and after such a campaign.

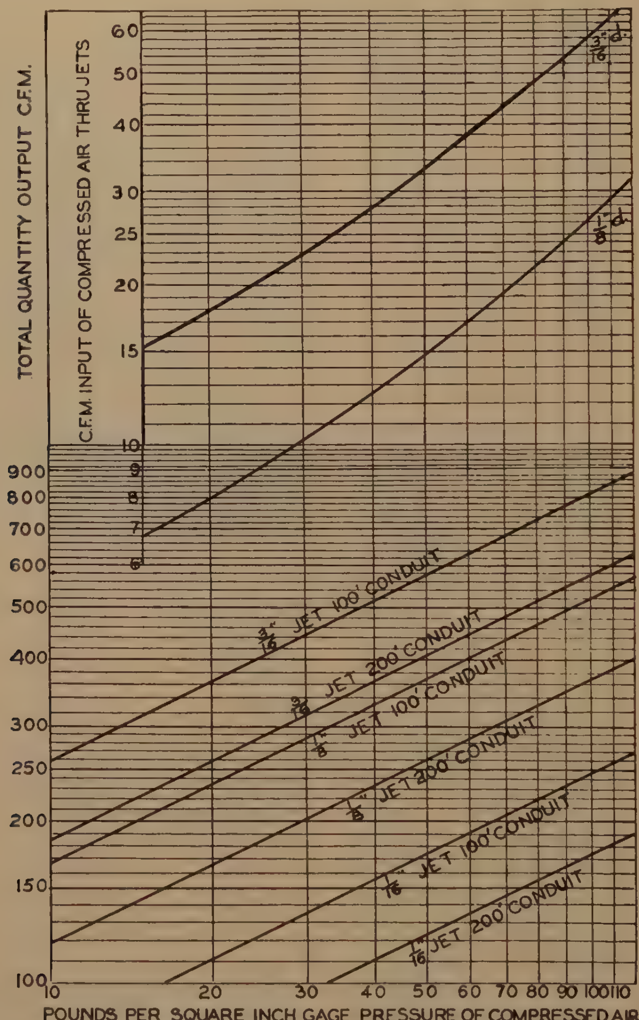

Fig. 6.—Results from tests with jets in plain 8-inch diameter conduits.

Conclusion

With rare exceptions, conditions in underground mines of all types can be greatly improved through a betterment of ventilation. This is especially true where there are many unusual problems, as in the Pennsylvania anthracite mines.

In operating its Panther Creek Valley mines, the Lehigh Navigation Coal Com-

pany Inc. must give close attention to ventilation. At present the company has two men engaged in such work, although both do not give their entire time to this assignment. The company, in an effort to provide more and better ventilation, and effect a general reduction in cost of production, has laid the groundwork for a long-range improvement program. This program includes the following items, which are either underway or under consideration:

1. Eventual replacement of all outmoded fans with fans that are new and efficient.

2. Revamping existing system where necessary, and using more detailed planning, to provide for larger capacity and greater permanency in main ventilating passages.

3. More detailed planning for conducting ventilation in new development and mining work.

4. Studies of (*a*) ventilation survey methods and records, (*b*) natural flow, especially gob drafts, (*c*) dust control, (*d*) utilization of compressed air, (*e*) materials and design of ventilating control devices, (*f*) effect of ventilation upon mine timber.

5. Systematic distribution of the essential facts thus obtained. (In this mining district this will be a problem of complexity, for each individual working place presents a separate ventilation problem and requires that the miners use special care to comply with the peculiar needs of their working place; thus the plan formulated must include all miners as well as the operating personnel.)

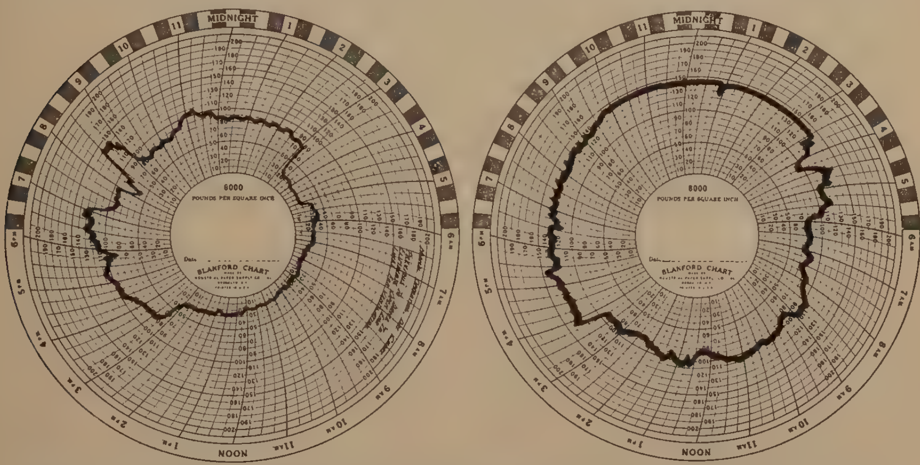

FIG. 7.—CHARTS OF COMPRESSED-AIR PRESSURE BEFORE AND AFTER BETTERMENT CAMPAIGN.

ACKNOWLEDGMENT

Grateful acknowledgment is due to the following members of the Lehigh Navigation Coal Company Inc.: Mr. D. C. Helms, Mining Engineer, and Mr. H. J. Earley, Superintendent, Coaldale District, for their much valued encouragement and guidance, and to Mr. L. A. LeVan, Assistant Mining Engineer, and Mr. E. J. Jones, Special Engineer, for valuable criticism, and suggestions.

DISCUSSION

(*A. Lee Barrett presiding*)

H. H. OTTO,* Scranton, Pa.—The paper presented by Mr. Beckwith is timely. The combination of heavy pitch and thick gaseous beds makes the ventilation problem unusually difficult, if the coal is to be won with a minimum loss of life. The Lehigh Navigation Coal Co. has

* Mining Engineer, Hudson Coal Co.

studied this problem through years, and apparently has solved it, although at a considerable expense.

The primary ventilation is an exhaust system with intake on the shaft and haulage roads and the return through various airways.

It is gratifying to know that the main return airways to the fan are to be on the same level as that of the haulage roads. This will make them easy of access, therefore they should be properly maintained and ensure the maximum ventilation at a minimum cost. This is an innovation in heavy-pitch coal mines.

Auxiliary ventilation also is being solved. Because of the heavy pitch, it is very difficult to use blower fans and the large spiral riveted pipe, therefore the use of compressed air in a very much smaller pipe with an injector at the end has been substituted.

It is extremely important that the face of the breasts be properly ventilated, and this apparently is being done, despite the fact that compressed air is an expensive method of ventilating a portion of a mine. The Company, however, is watching it very carefully and apparently has brought the costs within reason.

Some Problems in Connection with Ventilation of Mines Using Mechanical Loading Equipment

By A. W. Hesse,* Member A.I.M.E.

(New York Meeting, February 1941)

Ventilation of all types of coal mines is fundamentally the same, in that sufficient air must be provided to properly dilute and remove dangerous and obnoxious gases and leave the oxygen content of the air not under 19 per cent in all places where men and animals must work. Perhaps animals should not be mentioned, as the speed of mechanical loading leaves livestock behind and outmoded.

Since this paper has to do with the influence of mechanical loading equipment on ventilation, the problems will be confined to the splits and the coal-producing places of the mine.

The removal of coal dust from the air, or its rapid transmission to the return air courses; the possible increase in the liberation of gases by the multiplication of cuts in the coal made during one shift or multiple shifts; and the additional electrical equipment at the faces, are the factors that cause the added importance to good ventilation in the permissible equipment area of gassy bituminous coal mines.

Ventilation for Mechanical Loading in Gassy Mines

In gassy mines, the air currents must be maintained across the faces in such a way as to prevent short cuts to the returns and leaving pockets in which accumulation of gas might occur. To do this, two or more doors are required on almost every entry containing a track to the face, so located that in no event will a trip of cars block open the doors and permit the air to by-pass some place or places on the air-approach side. This method frequently requires the use of unusual lengths of duplex cables to reach the mechanical equipment at the faces, particularly where the distances between parallel working places are 100 ft., as at the Nemacolin mine. Fig. 1 shows a pillar section being worked with mechanical loaders. This layout requires carrying the air in on the right and releasing it on the left, because the track clearance is on the right side and the line brattice is built on this side from the last crosscut to the face. Thus, the air travels to the face with greater velocity and gives more assurance of sweeping out the gas that might be exuding from the crevices of the coal. All working places, in driving through solid coal, are thus treated alike. They are all potentially gassy and must be swept clear of whatever may contaminate the air. Once the air starts into the working section, it is not released until it does a good job. Of course the volume of air must be large enough not only to dilute the gases but to carry away explosive fumes and fine coal dust quickly.

Mechanical loading entails cutting and drilling of the coal during the same working period as the loading when two-shift and three-shift operations are employed, and all three processes create finely pulverized coal, which in itself is hazardous if not controlled by some means. If the ventilation is inadequate and gas accumulates, and a spark from defective electrical

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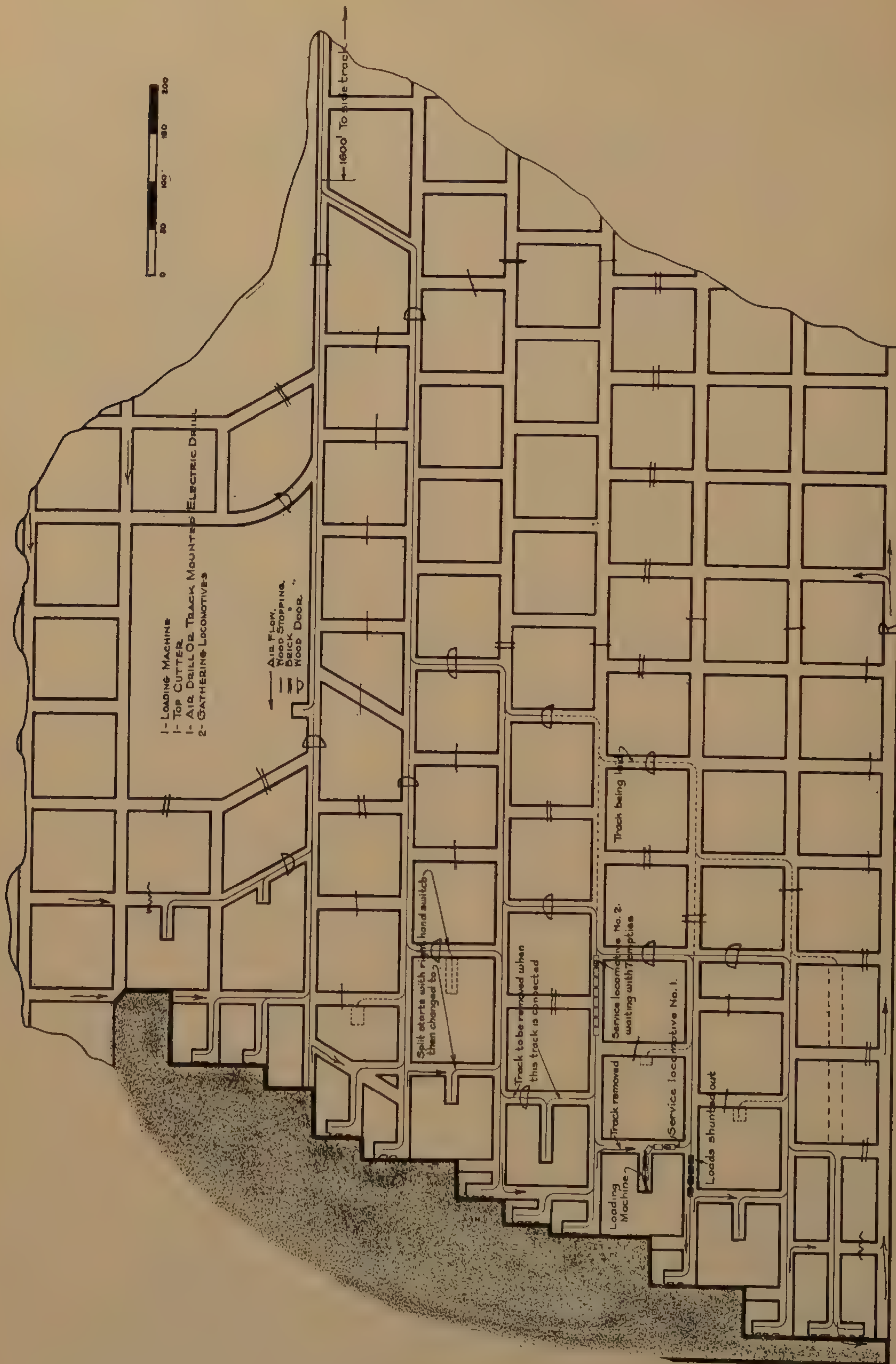

FIG. 1.—METHOD OF DRAWING PILLARS WITH MECHANICAL LOADERS.

equipment on the loader, cutter or drill ignites the gas, the coal dust propagates the flame. The problems then are:

1. To equip the cutting and loading machines with piping and water sprays and a water supply to wet down the dust during cutting and loading.

2. To provide the air volume and velocity to prevent accumulation of gas.

3. To maintain the mechanical equipment, so that the U. S. Government Permissible Equipment plate is kept inviolate.

Water for Laying Dust

Nemacolin chose long ago to lay water lines throughout the mine, so that every working place has a service line to which a hose, from the loading or cutting-machine spray, can be attached. The water is thrown onto the cutter bar of the cutting machine and showers the coal cuttings in such a way as to allay the dust effectively. With the loading machine, we have tried both the pipe spray on the loading boom and having the operator's helper spray the coal with the hose. Both means have been effective in keeping the air fairly clear of dust. This expedient not only provides better visibility and makes working conditions less hazardous, but reduces considerably the distances to which coal dust is carried into the return-air courses. In Pennsylvania, all entries must be rock-dusted, air courses as well as haulageways, so that samples taken every 500 ft. of "adhering and lodging" dust in developing entries shall contain not less than 55 per cent of incombustible material. Therefore, the less coal dust that is carried in the air, the less there will be deposited back in the air courses, and rock dusting will be less frequently required in the hardest places to get to, after the faces have moved on.

The water supply at Nemacolin is obtained by pumping the mine water to tanks at strategic points on the surface, from which the sprinkling water is drawn as needed. The excess mine discharge overflows the tanks to the streams.

Other means of water supply is by carrying a tank car, containing water, with the cutter or loader, from which it is pumped to the faces. This method is employed by a neighboring mine and avoids laying pipe lines; but, whether or not the saving in service lines is offset by the handling of extra equipment in an already congested area containing loader, mine cars, cutting machine, locomotives and their lengthy cables, I am not prepared to say.

Materials and Maintenance

Again referring to the traverse of air across these working faces, and observing the outbye track preparations on Fig. 1, it will be noted that double doors and stoppings are necessary before the double doors are removed from the inbye entries and crosscuts where the coal is being extracted. It is not the practice to use brattice cloth for stoppings to deflect the air. Wood or brick is employed, depending upon the importance of preventing air leaks, and cloth is used exclusively for line brattice to carry the air to the faces.

The laws of Pennsylvania demand the upkeep of permissible equipment to a standard, just as it was when approved by the U. S. Bureau of Mines. This means, of course, the parts of the machines that are liable to cause explosions. What has this to do with the ventilation situation? Because of the increased exposure to failures of the personnel to maintain an increased number of machines in service, the air must be kept free of explosive gases and fine coal dust, as contributory elements for an explosion setup.

Conveyor Mining with Pillar Retreat

Another situation arises where conveyor mining is used. Fig. 2 presents a case where brushing of roof must be done for the mechanical haulage and the room entries must be driven to the right, in order to

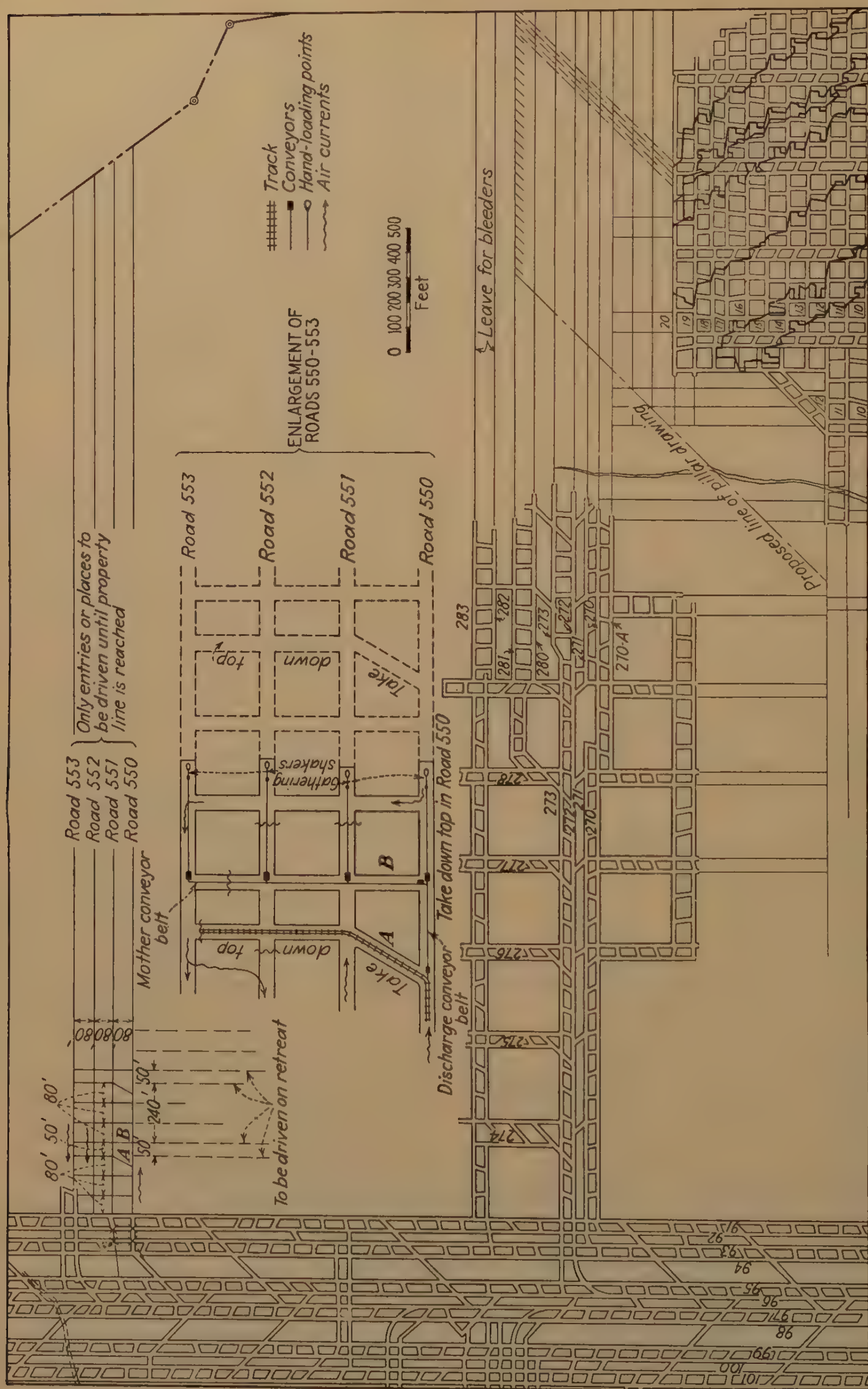

FIG. 2.—VENTILATION IN CONVEYOR MINING, DRAWING PILLARS.

have the pillar line (it is the intention to draw pillars on retreat) on an obtuse angle to the gob of previously worked-out territory. The air will travel roads 550 and 551 going in, enter the room entries to the right and return to the exhausting fan by way of the bleeders provided along the gob. Carbon dioxide comes off the old gob area, so it must be prevented from contaminating the development off roads 550. Therefore, any time a set of room entries cuts through into this area, the advancing air will tend to shortcut through the bleeders and push the CO_2 ahead of it. In the development of roads 550 to 553, inclusive, it would be desirable to place the haulage in road 553 and work to the right, as the cross entry would be lined up to drive on out to the right for the room entries. However, the return air must not cross over the trolley lines, to meet our standards, and it is not economical to provide a return airway parallel to and 80 ft. distant from road 550.

Also, it will be noticed that there are electric-motor drives at the end of each shaker conveyor, therefore the transmission lines to each unit must be rubber-covered, with high dielectric properties, and the motors must be explosion-proof, as the air, passing out the three working places on the right, finally passes over the motor on the left, or road 553, on its way out. For fear of getting these drives mixed, all the motors at the end of the shakers should be the same, permissible and interchangeable. The same applies to the motors on the air ducts to be used to clear the faces. These are necessary as the loading moves right along with the cutting and shooting and two to three cuts in a shift require rapid clearing of the atmosphere.

When these roads are driven to the property line, and entries started toward the old pillar section, the air will still be taken in on the right and out on the left until the room entries cut into the bleeders. Then the air should be split at the top, allowing just enough air to go left to keep the air courses clear and the major portion of the air carried up along the pillar line and out the bleeders remaining along the old gob line, to make sure no CO_2 is present when the next set of room entries cuts through. If these bleeders are not available, the air course of the room entries must carry the CO_2 and in working the entry stumps the men will be exposed to possible liberation while at work. Also it would be imperative that it be kept off the haulage road; therefore, a return would have to be provided on the right side or overcasts built to carry it over into the airways on the left at each room entry. In similar situations it might be methane instead of carbon dioxide.

Driving Entries Full Length before Cutting Rooms

Should it be impossible or uneconomical to draw pillars, a common method of working out the coal by conveyors is to drive the room entries to their full length before starting to drive rooms, then drive the rooms off one side only on the retreat, leaving a barrier along the air course (Fig. 3*a*). The air should be moved inbye with the advance of the belt, and the gases and smoke collected in the rooms should be removed to the return airway as quickly as possible. All shaker-conveyor and blower motors should be explosion-proof, because, since the practice is to leave the entry open, the gases are free to travel out of the rooms, each blower forcing the action in each individual room.

As the standing rooms increase in number, any continued liberation of gases must be prevented from increasing the flow over the electrical equipment; thus, the wisdom of causing the air to move away from the electrical equipment instead of over it, for cables will short and men will err.

In Fig. 3*b*, the problem is to drive the rooms on one side advancing, making only one setup for the "mother belt," and keep the electric coal equipment on intake air. The safe way is to drive three entries,

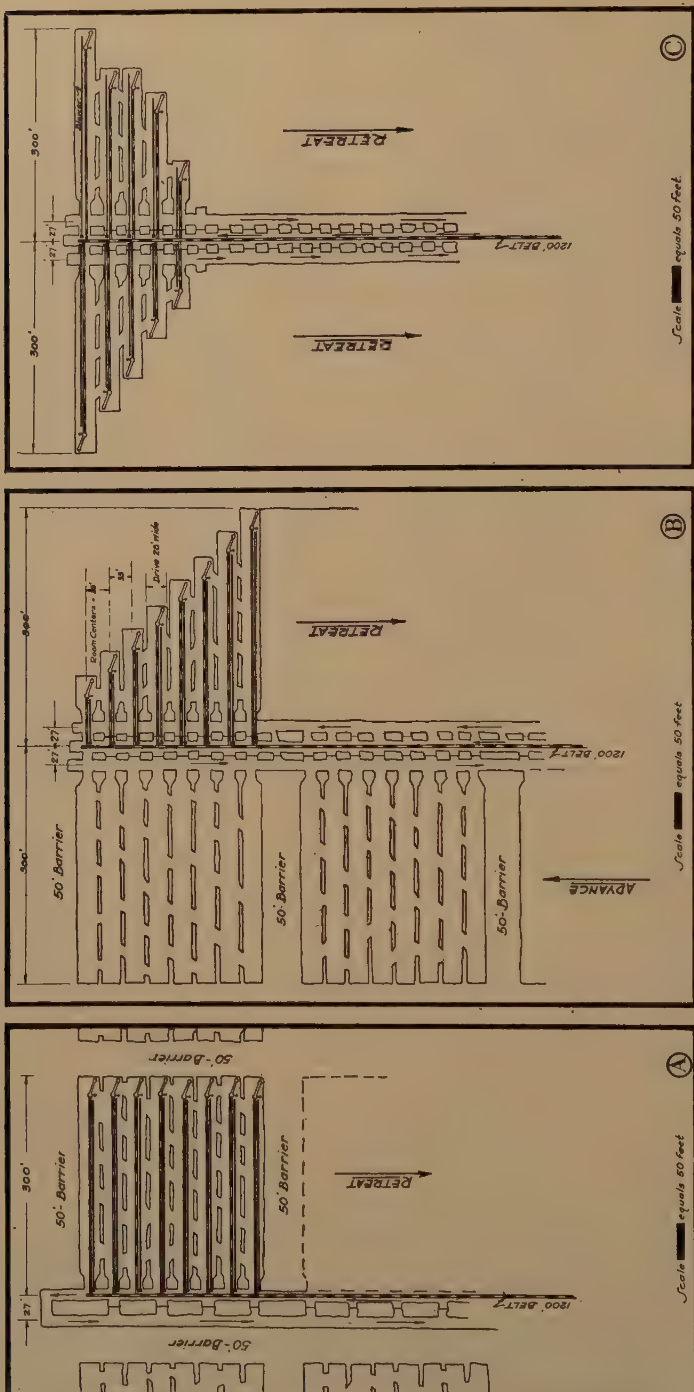

FIG. 3.—FOR CONVEYOR MINING, DRIVING ROOM ENTRIES FULL LENGTH BEFORE BEGINNING TO DRIVE ROOMS.

causing the air to flow inbye on the room entry as well as on the entry containing the "mother belt," letting it return the entry on the left, which keeps the accumulated gases behind the mine work and equipment. This could also be done with a two-entry system (Fig. 3*a*).

Fig. 3*c* plans driving the room entries to their full length before starting to drive rooms, then to drive the rooms off both sides on the retreat. This requires that the "mother belt" be carried the full length of the entries, using three parallel entries with the middle one for the belt and intake air. The problem is to get the air to the faces, particularly the last working rooms, since the shaking conveyors must cross the returns to get to the "mother belt," and if the crosscuts between outside and middle entries are not carefully sealed off around the shakers the air will have escaped into the returns before having ventilated the top end. In a gassy mine this is a serious situation. Since the potential between the intake and returns near the top end approaches zero, it requires little to hold the air on its course; however, the volume must be ample to allow for some escape through the brattice-cloth partitions, commonly used around the shakers, and depend upon the blowers and ducts to carry the air into the faces of the rooms.

Ventilation Practice

Quoting from U. S. Bureau of Mines *Information Circular* 7136: "It is evident that ventilation practice is not as efficient in many mines of the country as it should or could be. In too many mines it has been found that operating officials are ignorant of the proper ventilating requirements and do not realize or show proper respect for the hazard of gas."

This Circular calls attention to explosions having occurred during the year ending June 30, 1940, of which 15, it is thought, were initiated by gas and 4 by coal dust. Quoting further from this Circular: "In almost all the explosions initiated by gas it was indicated that the ventilating current had been interrupted and, in some of the mines, inspection for gas had been anything but adequate."

"Several of the ignitions were in so-called non-gassy mines, '*never before known to give off gas*,' or known to give off a 'little' gas or emit gas, 'occasionally.'

"One of the features of the gas ignitions was the fact that the blower fan-canvas tubing combination was a factor in at least 5 of the 15 explosions in which gas was ignited, usually in highly mechanized mines."

Because Government Approved explosion-proof equipment is used does not mean that the volume of air, its proper coursing, coal-dust removal and proper maintenance of permissible equipment should be ignored.

These are the problems that must be met in any kind of bituminous coal mining; but more particularly in mechanical operations.

DISCUSSION

(*John T. Ryan, Jr., presiding*)

E. McAuliffe,* Omaha, Neb.—Mr. Hesse has clearly stressed what have been advanced many times as the fundamental precautions that must be taken if coal-mine explosions are to be prevented; i.e., adequate and continuous ventilation, water on cutter bars of mining machines and for sprinkling during the cutting and loading process, as well as rock dusting to prevent the propagation of incipient explosions of gas or coal dust.

The problem remains to secure enforcement of methods on the part of operating officials and workers. Too often we overlook the fact that working faces are continuously moving away from the distribution devices set up, and it is in these new and numerous frontiers that gas is emitted in maximum quantity, and it is at these frontiers that explosions have their beginnings.

There is an old maxim that "what is everybody's business is nobody's business," and if enforcement of ventilation requirements is to be continuously maintained, some person, preferably a ventilation engineer, should have full charge of this important duty.

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* President, The Union Pacific Coal Co.

Effects of Underground Stopping Leakage upon Mine-fan Performance

By Raymond Mancha,* Member A.I.M.E.

(New York Meeting, February 1940)

When calculating the pressure-volume characteristics of projected mine-ventilating circuits by orthodox methods, certain basic assumptions are required in order to employ the various available empirical data. It is assumed, for example, that the mine air is an incompressible fluid subjected to isothermal flow, an assumption sufficiently accurate for practical purposes since pressure and temperature differentials are small throughout the average circuit. A more erroneous assumption, however, is that all air is accounted for as it travels throughout the mine.

No attempt is usually made to evaluate stopping leakage as it occurs; instead, the air volume required at the last crosscuts is assumed to enter the mine, travel the various intake air courses intact, sweep the workings and travel out by the return air courses to the point or points of exit from the mine. Actually, there is a leakage of air from intake to return at every stopping, the quantity of this leakage depending upon the tightness of the stopping and the pressure difference across the stopping.

Unfortunately, a lack of empirical data and knowledge of the condition of individual stoppings makes an exact analysis of underground stopping leakage impossible. Generally, leakage is most severe through the old stoppings outbye the circuit. These are also subjected to higher pressure differences than the newer inbye stoppings. Therefore the circuit air volume diminishes at a decreasing rate progressing from outbye to inbye the circuit.

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Consider a simple mine circuit consisting of one intake and one return air course, each of equal and uniform section area and interconnected at regular intervals with crosscuts equipped with stoppings. At any point X inbye the circuit the air volume q on the intake air course equals that on the return air course and decreases with diminishing rate from outbye to inbye the circuit between the limits of Q_1 completely outbye and Q completely inbye. Such a volume change is represented by curve a, Fig. 1, which is computed with the aid of the equation $q = Q_1 - (Q_1 - Q)(X \div L)^N$. This equation results from the general algebraic equation of form $q = k_1 - kX^N$ after solving for k_1 and k with the knowledge that completely outbye $X = 0$ and $q = Q_1$, also completely inbye $X = L$ and $q = Q$. Curves a, b and c, Fig. 1, show values of q as ordinates versus values of X as abscissas, for values of N equal to 0.50, 1.00 and 2.00, respectively.

Distribution of underground stopping leakage directly affects the air-volume distribution along the circuit, which in turn determines the circuit pressure gradient. To deliver any required air volume inbye a circuit or portion thereof requires a higher circuit-pressure differential if stoppings leak than is necessary with tight stoppings. Every mining engineer engaged in the analysis of projected mine-ventilation circuits is therefore confronted with the problem of making proper allowance for the effects of leaking stoppings.

The purpose of this paper is to present evidence in support of the author's suggestion that circuit pressure increase (per cent)

due to underground stopping leakage can be considered numerically equal to the leakage air volume expressed as percentage of air volume delivered inbye the circuit. Stated somewhat differently, pressure ratio P_1/P is equal to volume ratio Q_1/Q, wherein P_1 and P represent circuit pressure differential with and without leaking stoppings, respectively, and Q_1 and Q represent the air volume outbye and inbye the circuit, respectively, when stopping leakage is present.

An exact example will best serve to illustrate the use of the principle of pressure correction according to volume ratio as applied to a portion of a typical ventilating circuit. The accuracy of this principle depends upon the assumption of a constant coefficient of unit resistance as well as the assumption that the circuit volume if plotted will on the average lie within the confines of curves *a* and *b*, Fig. 1. If the first assumption is observed when applying the principle, the second assumption will be automatically satisfied in practically every case. Therefore, most accurate results can be expected when applying the principle to individual portions of the ventilating circuit, each consisting of a constant number of uniform intake or return airways separated from the adjacent return or intake airways by equally spaced crosscuts equipped with stoppings of the same general type of construction. The following example is an illustration.

Consider four parallel intake airways, each 4000 ft. long, 5 ft. high and 10 ft. wide. These intake airways form a portion of the intake circuit and lie parallel to a portion of the return circuit with interconnecting crosscuts uniformly spaced equipped with stoppings of similar but questionable construction. It is desired to deliver 100,000 cu. ft. of air per minute at the inbye end of that section of intake circuit in question. To do so, however, requires a volume input of 175,000 cu. ft. per min. at the outbye end with the difference in the two volumes leaking from intake to return along the circuit. Disregarding leakage, the problem resolves itself into one of passing 100,000 cu. ft. per min. a distance of 4000 ft. by means of four

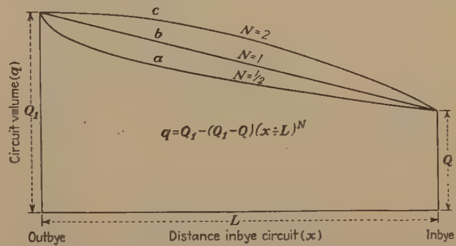

FIG. 1.—ONE INTAKE AND ONE RETURN AIR COURSE OF EQUAL AND UNIFORM SECTION AREA AND INTERCONNECTED AT REGULAR INTERVALS WITH CROSSCUTS EQUIPPED WITH STOPPINGS.

parallel intake air courses each of 50 sq. ft. section area (A) and 30 ft. perimeter (O). The resulting air velocity (V) in each entry is therefore 500 ft. per min. Assuming the well-worn friction coefficient k of numerical value 0.0000000217, the corresponding pressure drop is 2.50 in. water gauge when calculated by the accepted pressure formula $P = \frac{kLOV^2}{5.2A}$. Since normally leakage is disregarded, a pressure drop of 2.50 in. water gauge would be accepted as correct by most investigators.

When considering the effect upon the pressure resulting from the anticipated underground stopping leakage some observers have suggested that the leakage through all of the stoppings be considered equal, resulting in a circuit volume decreasing at a uniform rate along the circuit between the limits of 175,000 and 100,000 cu. ft. per min. This condition is suggested by this author as a limiting case only and is represented by curve *b*, Fig. 1. For a volume ratio of 175,000 ÷ 100,000, or 1.75, it will be seen from curve *b* of Fig. 2 that a pressure ratio of 1.94 results from this assumption. The predicted pressure drop based upon the assumption of uniform stopping leakage becomes $1.94 \times 2.50 = 4.85$ in. water gauge. In the writer's opinion, this

pressure is too high, because the assumption of uniform stopping leakage is unjustified by both theoretical and practical consideration. With stoppings of equal area and

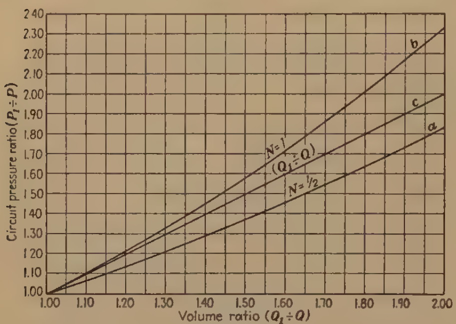

Fig. 2.—Circuit-pressure curves.

equal porosity uniform stopping leakage presupposes the same pressure differential across each stopping, which obviously is absurd. Furthermore, careful air-volume measurements taken at regular intervals along any underground circuit with leaky stoppings will always show a circuit volume decreasing at diminishing rate. When plotted along the circuit the circuit-volume curves will usually be found to lie between curves *a* and *b*, Fig. 1.

It is therefore the writer's opinion that the pressure ratio can be considered equal to the volume ratio with a high degree of accuracy when correcting pressure for the effect of underground stopping leakage. In the example at hand, which is a volume ratio of 1.75, the true pressure ratio should lie between the limits of 1.59 and 1.94, as shown by curves *a* and *b*, respectively, of Fig. 2. Assuming the pressure ratio equal to the volume ratio results in the selection of a pressure-ratio value of 1.75, which predicts a pressure of $1.75 \times 2.50 = 4.38$ in. water gauge across that portion of the circuit considered in this example. This value should be closer to the truth than that arrived at by any other method so far proposed.

The validity of the relationship $P_1/P = Q_1/Q$, finds mathematical as well as experimental support. For instance, again referring to the simple circuit described in paragraph 4, it is first assumed that all stoppings are completely tight.

The circuit pressure P therefore is equal to the sum of the incremental changes dp, in pressure difference from outbye to inbye the circuit or $P = \int_p^0 dp = -2kQ^2 \int_0^L dx$, wherein L is the circuit length from outbye to inbye. Integrating and substituting limits give the expression of circuit pressure, $P = 2kQ^2L$.

Again consider the same simple mine circuit previously described, with the exception that the stoppings will now be considered to leak air as occurs in actual practice. Traveling from outbye to inbye the circuit, at any point X, the pressure difference between intake and return air courses will decrease at the rate $dp/dx = -2kq^2$, wherein k is the circuit coefficient of unit resistance as before but q is the value of the circuit volume at X, varying between the limits of Q_1 completely outbye and Q completely inbye the circuits.

Again, the circuit pressure P_1 is equal to the sum of the incremental changes dp in pressure difference from outbye to inbye the circuit, or $p = \int_{p_1}^0 dp = -2k \int_0^L q^2\,dx$. After substituting for q its value in terms of X, integrating and inserting limits, the following formula for circuit pressure results:

$$P_1 = 2kL[Q_1^2 - 2Q_1(Q_1 - Q) \div (N + 1) + (Q_1 - Q)^2 \div (2N + 1)]$$

The same formula can be more conveniently expressed by making use of the volume ratio $Q_1 \div Q = R$ and substituting RQ for Q_1, which results in the formula for circuit pressure:

$$P_1 = 2kLQ^2[R^2 - 2R(R - 1) \div (N + 1) + (R - 1)^2 \div (2N + 1)]$$

Circuit-pressure ratio, with and without stopping leakage, is shown by the formula:

$$P_1 \div P = R^2 - 2R(R - 1) \div (N + 1) + (R - 1)^2 \div (2N + 1).$$

Curves *a* and *b*, Fig. 2, plot values of circuit-pressure ratio, $P_1 \div P$, as ordinates versus volume ratio, $R = Q_1 \div Q$, as abscissas for values of $N = 0.50$ and $N = 1.00$, respectively.

The assumption of values for N between the limits of 0.50 and 1.00 is in close agreement with practical observation. Furthermore, the curves in Fig. 1 show that n must have some value less than unity if the circuit volume decreases with diminishing rate from outbye to inbye.

Therefore, since it is impossible to evaluate N more accurately than to ascribe limits thereto of 0.50 and unity, the relation of the *c* curve to the *a* and *b* curves of Fig. 2 suggests that the use of this *c* curve is an accurate and useful means of correcting circuit pressure for the effects of underground stopping leakage.

DISCUSSION

(*W. H. Lesser presiding*)

R. D. HALL,* New York, N. Y.—Mr. Mancha seems to express the belief that leakage in a mine is mostly through stoppings. Has he given consideration to the probability of extensive leakage through the pillars themselves, to say nothing of leakage to the surface?

After the explosion at the Horning mine, near Pittsburgh, Pa., the fire by which it was caused was carefully sealed and the seals were closely watched and kept tight. The fire continued to burn, however, until, by a rearrangement of the air currents, the water gauge was made about equal on each side of the fire area. This tends to show that the fire was not kept alive by the water gauges created by its own combustion but by the difference of air pressure in the surrounding air or between the mine air and the air at the surface above the mine into which the mine air could leak.

Water will pass through rock and coal measures, and often it can be seen entering through the coal. Why should not air do the same, after the water in the measures has been spent, leaving its former channels open to the passage of air? Though the air has not the same pressure as the water, it has far greater permeability. Dr. George H. Ashley, in speaking about the strength of water barriers, has called attention to the long distance water will travel in unbreached pillars, and C. H. Tarleton has given evidence of the passage of methane and ethane for the distance of ½ mile through solid coal from a capped well to mine workings, as testified by the presence of the ethane in these workings, which, as is well known, is rare in American mines. In the mines of the Pocahontas Fuel Co., holes drilled into the coal wherever water did not intervene largely ridded the coal of methane over a large area, exhibiting the permeability of some coal, at least.

In Great Britain, it has been stated that rock dust closes the crevices by which air passes through pillars. The face of rock-dusted ribs becomes coated with gypsum, melanterite and ferrous or ferric sulphate from the pyrite in the coal and the calcium in the limestone, if limestone is used for rock-dusting.

Moreover, shooting makes crevices in pillars. J. F. Joy asserts that his method of sawing coal and prying the blocks loose with a hydraulic pad gives satisfactory results only after it has been used for a distance of 15 ft. from a shot face, because for that distance the coal is badly creviced; hence it is likely that 30 ft. if not more of any pillar, 15 ft. on each rib, is opened to air, and that channels not thus opened, but original and adequate, will permit the air to travel the rest of the way.

Furthermore, the pyrite in the coal turns to sulphate with immense expansion, which rends the pillars. The clay under the coal flows and pulls the pillars apart and, by letting them down, induces crevicing. Limestone in the floor has a similar expansive effect and it may also cause expansions in the coal itself. Crevicing from weight is a further way in which air channels may be enlarged or expanded. In subbituminous and lignitious coals, the coal dries; hence, crevices are large and numerous. Perhaps the reason why rock dust is helpful in preventing leakage in Great Britain is because weight has more often creviced the pillars. In the Nova Scotia mines, the pillars move into the openings under pressure, and the Belgians sometimes have reason to say *La veine est marché* (the bed has moved forward).

* Engineering Editor, *Coal Age.*

As Mr. Mancha says, it is a universal assumption in calculating formulas that the flow is isothermal and the air incompressible but, as he recognizes, it is erroneous. I question whether it is less erroneous than the assumption Mr. Mancha condemns and would correct, but it is one less easy to generalize and revise.

It will be noted that "L is the circuit length from outbye to inbye" and not, as is more customary, the length of the entire circuit. With a current that does not make a circuit there may be no leakage (except to the surface) and the formula will follow the textbooks.

R. Mancha (author's reply).—I am aware that there is other leakage to be considered besides "stopping" leakage, but the subject of my paper was the effects of stopping leakage only upon the circuit pressure-volume characteristics. In short, my paper was limited to the effects of a particular form of leakage rather than a general discussion of all forms of leakage.

Pitot-tube Field Tests of Axial-flow Mine Fans

By Raymond Mancha,* Member A.I.M.E.

(New York Meeting, February 1942)

A test of any fan requires the determination of such data as fan pressure, air volume handled by the fan, and power input to the fan shaft.

When testing operating mine fans of the centrifugal type, test methods have been employed, for the sake of convenience, that permit great latitude in detail of procedure resulting in wide variance in results by different investigators.

Some controversial aspects of field fan testing are the result of past failure to define clearly the various forms of fan pressure and to a lack of understanding of the accurate procedure for the measurement thereof. Also, the anemometer for air-volume measurement, although convenient, if carelessly used and maintained is subject to a wide error range with variation in method of application and in instrument accuracy.

The appearance of the axial-flow type of mine fan offers the possibility of refinements in field testing that were impractical with the centrifugal fan. First of all, the fan inlet and outlet are more sharply defined, giving closer agreement as to just where the fan begins and ends; thus there is less disagreement as to the proper location for measurements of fan pressure.

The area of passage into the inlet of an axial-flow mine fan usually constricts, causing high air velocities with accompanying steadiness of flow. Thus the fan inlet offers an ideal location for accurate air-volume measurements with the Pitot tube, the proper use of which is well established by code. Thereby the controversial aspects of such measurements are eliminated. Use of the Pitot tube also facilitates a more accurate measure of fan pressure, since the individual pressures measured can be properly weighted, as will be explained later.

Determination of power input to the axial-flow mine fan operating in the field is subject to the same inaccuracies as with the centrifugal fan if the axial-flow fan is also belt driven. However, because of the higher rates of rotation usual with axial-flow fans, direct-driven units are more numerous and it is unnecessary to assume a drive efficiency, thus eliminating one possible source of error.

It is the purpose of this paper to define and explain the various types of pressure measurements involved in a field test of an axial-flow mine fan, and to further explain the proper use of the Pitot tube when measuring pressures and volumes.

Definition of Fan Pressures

The two fan pressures most commonly discussed are fan total and fan static pressure, each of which has the same meaning regardless of whether the fan is blowing, exhausting, or blowing and exhausting, as in the case of a fan with ducts connected to both outlet and inlet.

Fan total pressure is the amount by which the total pressure of the air delivered by the fan exceeds the total pressure of the air received by the fan. This pressure is a measure of the maximum amount of the work done on the air by the fan that is available for external use.

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* Manager Ventilation Division, The Jeffrey Manufacturing Co., Columbus, Ohio.

For a blowing fan, fan total pressure is the total gauge pressure obtained by traversing the fan outlet with a facing Pitot tube connected by a hose to one leg of a manometer of which the other leg is open to the atmosphere.

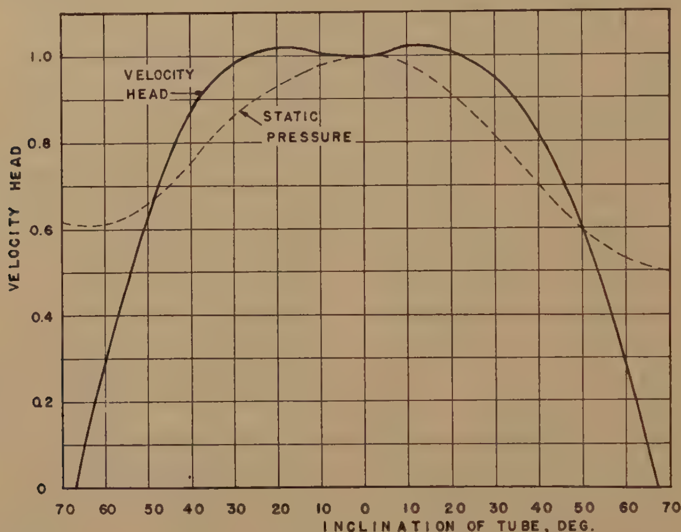


FIG. 1.—PITOT TUBE WITH STATIC HOLES AT SIDES, TOP AND BOTTOM.
Data obtained from *Transactions* of the American Society of Mechanical Engineers.

For an exhaust fan, fan total pressure is the algebraic difference between the total gauge pressures obtained by traversing both the fan outlet and inlet with a facing Pitot tube connected by hose to one leg of a manometer of which the other leg is open to the atmosphere.

With a fan both blowing and exhausting (such as a fan underground), fan total pressure is the algebraic difference between the total pressures obtained by traversing both the fan outlet and inlet with a facing Pitot tube connected by hose to one leg of a manometer of which the other leg is exposed to the same pressure during each traverse.

Fan static pressure is the amount by which the static pressure of the air delivered by the fan exceeds the total pressure of the air received by the fan. This pressure is a measure of the minimum amount of the work done on the air by the fan that is available for external use.

With a blowing fan, fan static pressure is the static gauge pressure obtained by traversing the fan outlet with a static Pitot tube connected by a hose to one leg of a manometer of which the other leg is open to the atmosphere.

For an exhaust fan, fan static pressure is the algebraic difference between the static gauge pressure at the fan outlet and the total gauge pressure at the fan inlet as obtained by traversing the fan outlet and inlet with a static and facing Pitot tube, respectively, connected by a hose to one leg of a manometer of which the other leg is left open to the atmosphere.

With the fan installed below ground, the fan static pressure is the algebraic difference between the static pressure at the fan outlet and the total pressure at the fan inlet as obtained by traversing the fan outlet and inlet with a static and facing Pitot tube, respectively, connected by a hose to one leg of a manometer of which the other leg is exposed to the same pressure during each traverse.

Mine ventilating pressure is the irreducible minimum pressure across a mine required to maintain the desired

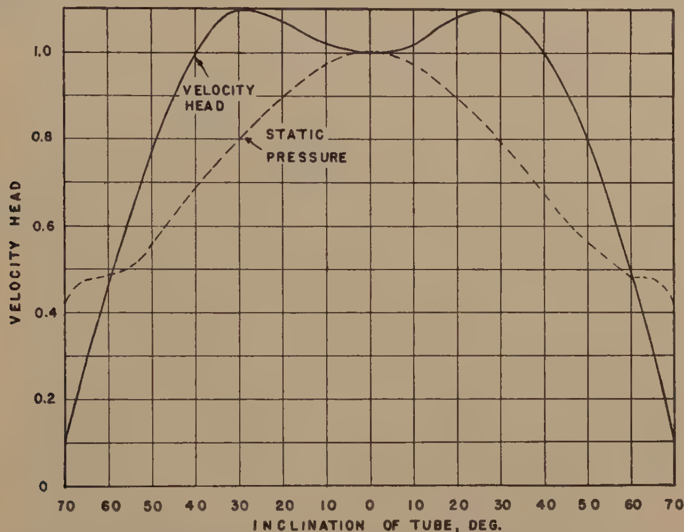


FIG. 2.—BRABBEE TUBE WITH STATIC ORIFICES AT THE EXTREMITIES OF TWO DIAGONALS.
Data obtained from A.S.M.E. *Transactions.*

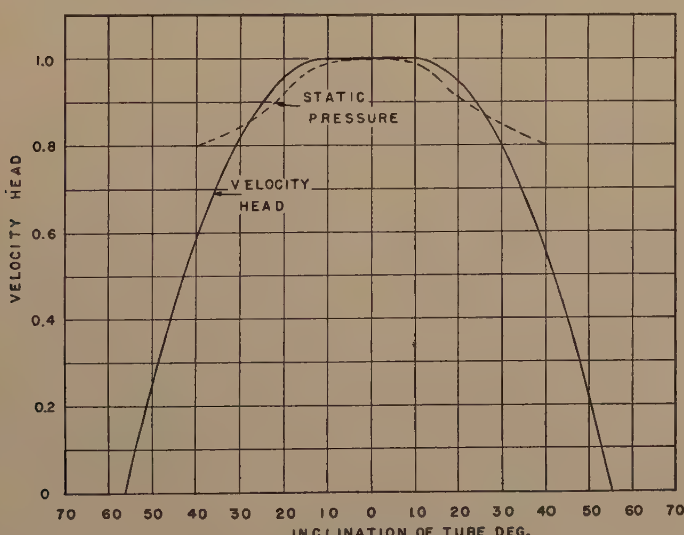


FIG. 3.—PRANDTL TUBE.
Data obtained from A.S.M.E. *Transactions.*

ventilation. Regardless of the fan location, mine ventilating pressure accounts for all underground pressure losses as well as the loss in the air at the mine exit. With a blowing or underground fan, the exit loss generally consists of velocity pressure only. With an exhaust fan installation, however, the exit loss, if any, is limited to the type of loss associated with changing the direction of air flow at a bend.

For a mine ventilated by a blowing fan, mine ventilating pressure is the total gauge pressure measured just inside the

mine entrance, as at the air-shaft collar or drift mouth. With the fan exhausting, mine ventilating pressure is the total gauge pressure measured just outside the mine is obviously the logical reference pressure whenever the ventilating efficiency of a mine fan is considered. Fan total pressure is an unsuitable reference because alge-

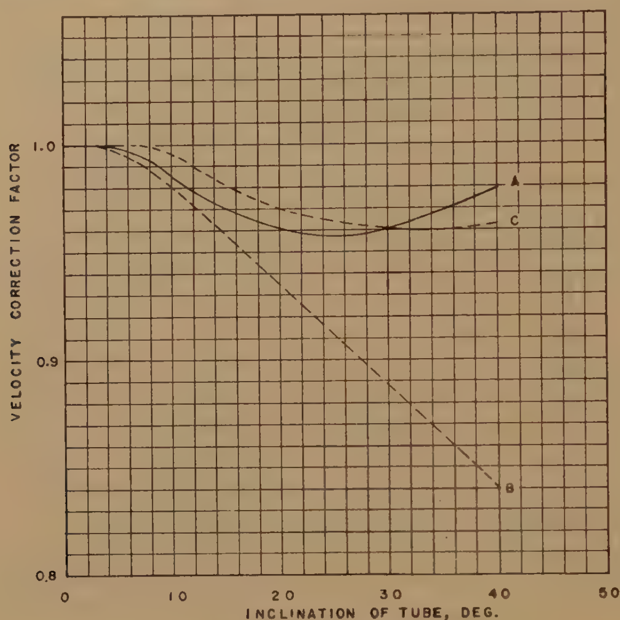


FIG. 4.—CORRECTION FACTORS.
A. Total correction factor for Prandtl tube.
B. Total correction factor for Pitot tube with static orifices at sides, top and bottom.
C. Total correction factor for A.S.H. and V.E. tube.
Data obtained from A.S.M.E. *Transactions*.

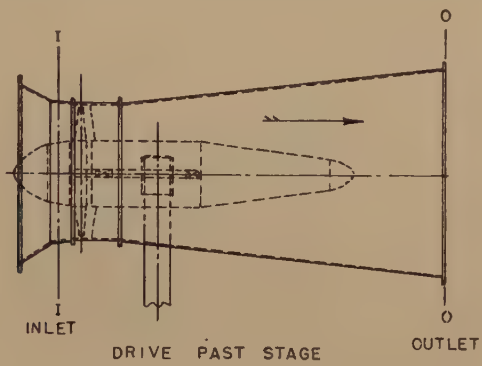
exit in front of the fan. With the fan below ground, mine ventilating pressure is the total pressure difference in the air just ahead of and behind the fan.

Usually, the mine entrance, exit or airway, exceeds in section area the inlet or outlet of a propeller mine fan. Therefore, regardless of the fan location, the air approaching the fan can be expected to undergo accelerated flow, whereas the air leaving the fan will be subjected to retarded flow. The efficiency of the former process justifies neglect of the accompanying pressure losses; however, the latter process—namely, retarded flow—involves pressure losses that must be accounted for.

The fan pressure most closely corresponding to the mine ventilating pressure braically it exceeds the mine ventilating pressure by an amount equal to the retarded flow-pressure losses following the fan outlet and, if the fan is blowing, by the bend loss in the air-shaft hood as well. The possible magnitude of this discrepancy is best illustrated by considering the effect of eliminating the fan diffuser or stack. On the other hand, fan static pressure can be safely equated to the mine ventilating pressure because in this way the fan outlet velocity pressure is available to compensate for the other losses mentioned.

For field-test purposes, therefore credit a blowing or underground mine fan with the true fan static pressure measured as already described. The exhaust fan when field-tested should be credited with the

total gauge pressure measured by a facing Pitot tube traverse of the fan inlet. This pressure usually closely approximates fan static pressure and equals it exactly for opening coincides with the point in question, and so that the tube points directly into the resultant air velocity at the point of measurement. Any angularity between



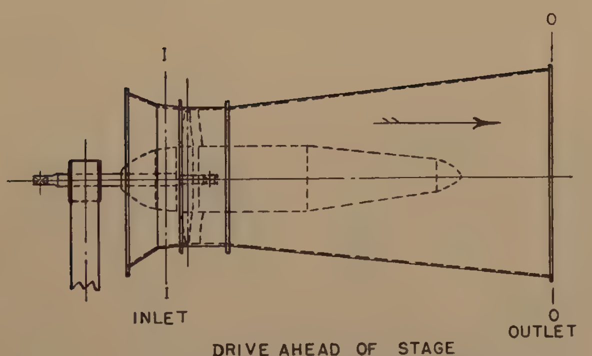


FIG. 5.—PROPER LOCATION FOR PITOT TRAVERSE PLANES.

the special case of air flow past the fan outlet entirely free from nonaxial velocity and nonaxial acceleration.

PRESSURE MEASUREMENTS WITH PITOT TUBES

Air-pressure measurements are accurately and easily performed with the Pitot tube and manometer.

When true total pressure is measured at any point in an air stream, the Pitot tube should be directed so that the total pressure the tube and the resultant air velocity is known as an angle of yaw, and has the effect of making the total pressure gauge read low. Figs. 1, 2 and 3 show the effects of yaw upon three different kinds of tubes when static or velocity pressure is being measured.

It is common practice to consider total pressure as the algebraic sum of the measured static pressure and the computed velocity pressure referred to the mean axial velocity normal to the plane in which

the static pressure is measured. The accuracy of this assumption depends upon the uniformity of the axial velocities over the plane, as well as freedom from nonaxial velocities.

When measuring static pressure at any desired point in an air stream, the Pitot tube should be placed with the static pressure holes in a plane that includes the point of measurement and is normal to the resultant air velocity at that point.

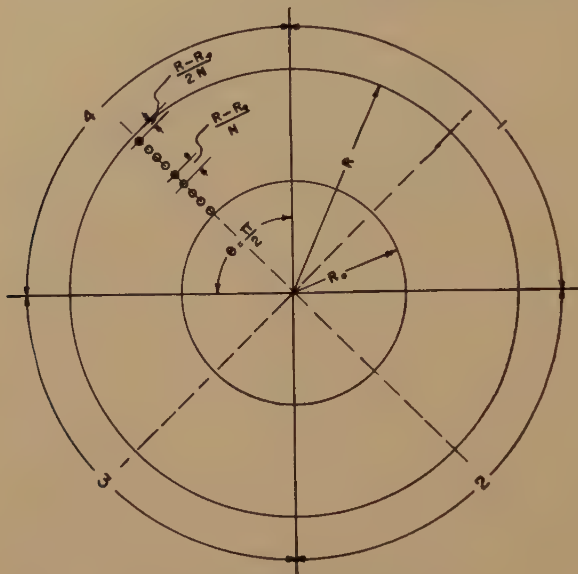


FIG. 6.—TRAVERSE PLANE.

Nonuniform axial flow introduces error because the mean of the squared axial velocities exceeds the mean axial velocity squared. Nonaxial flow produces error because the nonaxial velocity pressure goes unaccounted for. In either case the error leads to an apparent total pressure that is too high or too low, depending upon whether the static pressure is negative or positive.

When determining the total pressure in the air immediately following a bend, or at the outlet of a fan, or at similar places, it is especially important that a complete total pressure traverse of the section be made with a facing Pitot tube. To ensure accuracy, the traverse is always essential unless it has been previously ascertained that uniform pure axial flow exists only across the plane of total pressure measurement.

The effect of yaw upon static-pressure measurement is shown by Figs. 1, 2 and 3.

Uniform static pressure over a plane of measurement is possible only when the air flow past the plane is purely axial and of either uniform or nonuniform velocity distribution. The static-pressure distribution varies in value whenever nonaxial accelerations or velocities exist in the plane of static pressure measurement. Therefore, to obtain accurate measurements of static pressure at the outlet of any fan it is usually necessary to traverse the outlet with a static Pitot tube.

Total or static-pressure measurements at the inlet or outlet of a high-pressure propeller fan usually can be obtained accurately with a Pitot tube held parallel to the longitudinal fan axis, thereby neglecting the effect of yaw. This is so because the inlet plane of pressure measurement is bounded by almost parallel sides

and is either in or immediately following a zone of axially accelerated air flow. The outlet plane of pressure measurement, although following a zone of gradually retarded axial air flow, is fairly free from nonaxial air velocities because of the action of properly designed guide or straightener vanes.

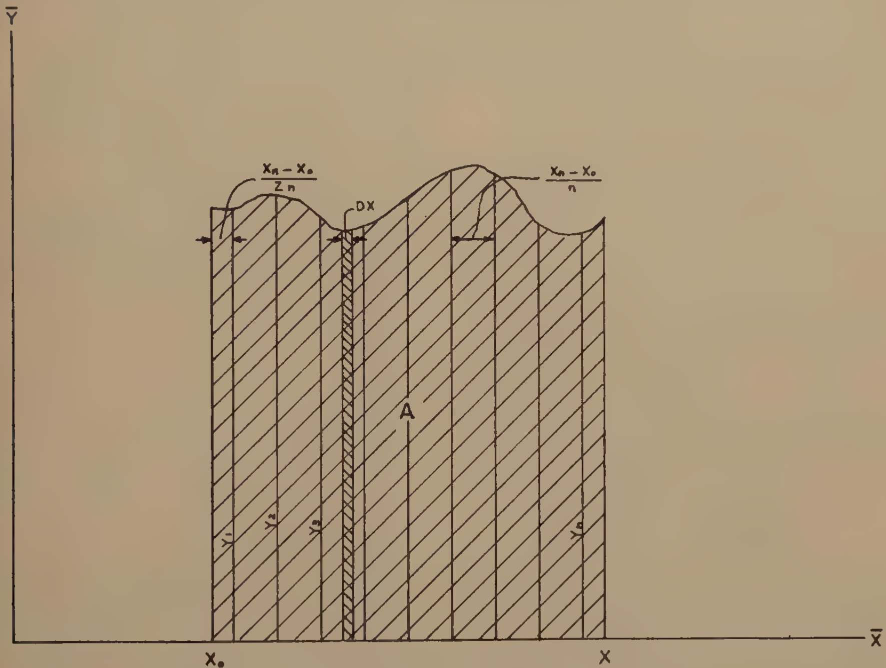


FIG. 7.—CALCULATIONS FOR VOLUME AND PRESSURE.

AIR-VOLUME MEASUREMENTS WITH PITOT TUBES

Measurements of velocity pressure serve as a basis for computation of air volume and therefore are just as essential as measurements of fan pressure. The air velocity v at any point of measurement is expressed as $v = 1097.5\sqrt{\frac{VP}{d}}$, in which VP represents the measured velocity pressure (inches water) and d is the specific weight of the air (lb. per cu. ft.). The velocity v therefore is expressed as feet per minute.

When measuring velocity pressure, the Pitot tube should be placed with the static-pressure holes in a plane that includes the point of desired velocity measurement and that is normal to the resultant air velocity at that point.

Actually, the Pitot tube measures the difference between the total pressure at the front of the tube and the static pressure at the static holes. The total pressure difference between the two points is negligible, as is likewise the change in direction of air flow. Therefore, the velocity pressure as measured with the Pitot tube is the resultant velocity pressure that exists in the plane of the static-pressure holes.

Similarly, the section area used to compute the air volume should be equal to the section area in the plane of the static holes, normal to the fan axis. The significant air velocity accordingly is the axial component of the resultant velocity measured as described above.

Fig. 4 shows the amount of error with each of three types of Pitot tubes arising

from holding the tube parallel to the fan axis with the static holes in the plane of desired velocity measurement, and considering the velocity thus measured as the significant velocity. Such slight errors, when using either the Prandtl, or A.S.H. & V.E. tubes, justify this procedure during field tests.

Selection of Traverse Plane

At a location for pressure or volume measurements, the following conditions are desirable: (1) pure axial air flow, (2) uniform velocities at equal radii, (3) freedom from pulsations, (4) traverse on radial lines located at least three diameters distance from any obstruction in the plane of pressure measurement 10 diameters distance from any obstruction directly ahead, and three diameters distance from any obstruction immediately behind the radial traverse line. Seldom is it possible to strictly observe all of these desirable features, but the location that most nearly satisfies the desired conditions should be selected.

Air volume is measured preferably immediately ahead of the fan rotor, where the air is in accelerated flow and free from turbulence. When testing a pressure-type (blowing) mine fan, however, a period should be selected when there is little or no outside wind. Otherwise the inlet velocities will vary with time.

Fig. 5 shows general arrangement of fan with drive located past and ahead of the fan stage (rotor and vanes). The planes 1-1 are ideally suited for measurements of either volume or pressure at the fan inlet. The planes O-O correspond to the fan outlet at which it may be desired to measure total or static pressure.

System of Traverse

A suitable plane having been selected for pressure or volume measurement the plane should be divided into at least four equal-angle sectors and the bisector of each sector should be traversed for pressure, velocity, or both, as is desired. Fig. 6 shows such a traverse plane divided into four sectors and with N equally spaced stations selected along each traverse line. As a rule, it is good practice to select at least eight stations on each traverse line between the outer radius R and the inner radius R_0. The spacing of these stations should be determined as shown by Fig. 6.

Obviously, the pressure or velocity reading thus obtained at a particular radius on a particular traverse line represents the average value at that radius over the entire angle θ included within the sector.

Calculations for Volume and Pressure

The area under any curve such as A in Fig. 7 is equal to $\int_{X_0}^{X} y\,dx$, which in turn very nearly equals $\left(\frac{y_1 + y_2 \cdots y_n}{n}\right)$ $(X - x_0)$ if n is sufficiently large.

The air volume q passing any sector of angle θ radians between radii R and R_0 is expressed by the equation $q = \int_{R_0}^{R} vr\theta dr$ in which v is the air velocity at radius r. Substituting q for A, R for X and $vr\theta$ for y, the following expression results:

$$q = \theta\left(\frac{v_1R_1 + v_2R_2 \cdots v_nR_n}{n}\right)(R - R_0)$$

By averaging the velocities in the different sectors at the same radius and expressing this average velocity as V, the final expression for the air volume passing the entire plane of traverse becomes:

$$Q = 2\pi\left(\frac{V_1R_1 + V_2R_2 \cdots V_nR_n}{n}\right)(R - R_0)$$

The rate of energy transfer w by the air passing a traverse section of angle θ radians between radii R and R_0 is expressed by the equation

$$w = \int_{R_0}^{R} p\,dq = \int_{R_0}^{R} pvr\theta dr,$$

in which p and v represent air pressure and velocity at radius r. Substituting w for A, R for X and $pvr\theta$ for y, we have the expression

$$w = \theta \left(\frac{p_1v_1r_1 + p_2v_2r_2 \cdots p_nv_nr_n}{n}\right)(R - R_0)$$

By again substituting the average values, in the different sectors, of pressure P and velocity V at each radius traversed R, the final expression results for the rate of energy transfer W by the total air volume Q passing the entire plane of section traverse:

$$W = 2\pi \left(\frac{P_1V_1R_1 + P_2V_2R_2 \cdots P_nV_nR_n}{n}\right)(R - R_0)$$

From the relation $W = PQ$ follows the equation

$$P = \frac{W}{Q}$$

$$= \frac{2\pi}{Q} \left(\frac{P_1V_1R_1 + P_2V_2R_2 \cdots P_nV_nR_n}{n}\right)(R - R_0)$$

which is the general expression for the average pressure P at the plane of traverse.

Therefore, when measuring average pressure at any traverse plane it is necessary, for the purpose of proper weighting, that velocities be measured in addition to and simultaneously with the individual pressures.

Power Determination

The mechanical power delivered to the fan shaft is less than the electric power input to the motor terminals by an amount equal to the motor losses and drive losses.

The power input to the motor terminals should be measured directly with an indicating wattmeter or with a recording watt or watthour meter. It is bad practice to measure current and voltage and then attempt a "guess" at the power factor of an alternating-current motor.

The motor losses can be most closely approximated by the motor manufacturer upon receipt of such information as the power input to the motor, the voltage between the different phases and the current flow per phase.

If the fan is driven directly connected to the motor, the motor output can be considered equal to the fan input, thereby neglecting any possible loss in the coupling. If, however, the fan is driven through one or more drives it is necessary to deduct an estimated drive loss.

Flat-belt or V-belt drives with short centers are assumed to be 95 per cent efficient. The true efficiency varies with such factors as belt tension, alignment, load. However, practice has justified the efficiency assumption of 95 per cent, or a power loss of 5 per cent for each pair of pulleys required to drive the fan.

Conclusions

The careful application of the various procedures suggested herein will produce field-test results that are in close agreement with factory fan tests of laboratory precision. This is particularly true of the pressure-volume measurements and applies to a lesser degree to the power measurements, which involve efficiency estimates for motor and drive. However, most of the serious sources of error to which field fan tests have been subjected in the past are hereby eliminated or greatly reduced in magnitude.

DISCUSSION

(E. R. Kaiser presiding)

C. M. Smith,* Washington, D. C.—In making recent acceptance tests of high-capacity underground fans, the practice we had previously followed in such situations of determining the rate of flow by anemometer was abandoned in preference of Pitot-tube testing, because of difficulties that would have been encountered in attempting to establish satis-

* Editor, *Mechanization.*

factory measuring sections for anemometer traversing. One fan had been set just to one side of the top of a raise, so that its incoming current made an abrupt right-angle turn at the fan intake. On the discharge side the passage expanded rapidly and irregularly, then divided a short distance from the fan, so that there was no place where a satisfactory anemometer traversing section could be established on either side of the fan. Both fans were of the drive-past-stage type generally similar to that shown in the upper diagram of Fig. 5. In each case, the traverse section chosen was downstream from the drive, at the end of the cylindrical portion of the center housing that covers the drive shaft and bearings. Objections can be raised to selection of this section, but it seemed preferable to the outlet section from the standpoint of air flow, particularly because preliminary readings in the outlet section gave comparatively low velocity pressures and their variability indicated appreciable turbulence.

The inlet section (I-I in Fig. 5 of Mr. Mancha's paper) was inaccessible for Pitot-tube traversing because of the presence of a concrete bulkhead that serves as the fan partition. Even if section I-I had been available, we might have backed away from it, because of the 90° turn directly ahead of the fan.

An attempt was made to compensate for the existence of the drive-belt housing a short distance upstream from the test section, by traversing on as many as 10 radii in the section. Points along the radii were chosen by the method of equal annular areas, rather than uniformly as suggested by Mr. Mancha. While the presence of the drive-belt housing is a disadvantage, the situation we were faced with was, in some respects, comparable with that which exists in section I-I when the drive is ahead of the stage, as shown in the lower portion of Fig. 5.

Although the author's recommended section I-I could have been used at one fan, we were fearful that the Pitot tube might be unwieldy in the rapid air current and might accidentally be drawn into the fan blades, with possible damage to tube and blades. However, experience in traversing at the section that was finally chosen indicated that this fear was unsound, as the tube was easily handled despite the fact that many traverse points showed air speed well in excess of a mile a minute.

Test conditions were by no means ideal but results were satisfactory, inasmuch as they indicated that the fans were performing well within the manufacturer's guarantee, therefore they were used as a basis for accepting the fans. Where a Pitot tube of standard design and a sensitive manometer, one on which pressures can be estimated to a thousandth of an inch of water, are available, and a velocity in excess of a thousand feet per minute prevails throughout the traverse section, as is usual in high-capacity propeller-type mine fans, Pitot-tube traversing should be more accurate than anemometer traversing and almost as expeditious.

An Investigation of Dust Suppression in the Pittsburgh Seam

By D. H. Davis,* Junior Member A.I.M.E., and G. R. Gardner†

(New York Meeting, February 1942)

Increasing realization of the importance of dust control, and the recommendations of various agencies, have led the mining industry to become actively concerned with this problem. The background and necessity for adequate dust-suppression measures have been discussed in a number of publications, including those of the U. S. Bureau of Mines, and the use of spray water to reduce dust at the face and on roadways has been recommended. The problem resolves itself into obtaining adequate dust reduction at a reasonable cost. Inasmuch as the term "adequate dust suppression" has never been clearly defined, nor have permissible limits of dustiness (in terms of dust count) been established, it is apparent that any method selected may be required to meet additional demands. Until the industry has been given some indication of permissible limits of dustiness, no permanent solution is possible. Progress will be somewhat handicapped until such information is available.

In addition, dust reduction must be obtained without the addition of any excessive moisture to the coal, since consideration must often be given to:

1. Quality standards, particularly reduction in heat value.
2. Handling characteristics, particularly in regard to unloading from railroad cars and bins.
3. Application of dry methods of coal cleaning.
4. Necessity for fine screening within specified limits of oversize and undersize.

Little detailed information is available as to the advantages and disadvantages of the several methods of water distribution, the location and protection of sprays, operating data on pressure, quantity of water, and so forth, and results obtained. In 1940 an investigation was undertaken to determine and evaluate the principal factors involved. This paper constitutes a progress report on this work and cannot in any way be regarded as complete on any one of the several phases.

Classification of Dust Sources

The various sources of underground dust may be classified as primary or secondary. A primary source is one associated with appreciable degradation of size, that is, where the greater part of the actual fines capable of becoming air-borne dust are produced. A secondary source may be considered as one in which the greater part of the fines have been produced at some previous operation and are in such condition that they can be dispersed as air-borne dust. These conceptions appear to be of practical significance, as experience indicates that when dust is controlled at the primary sources the importance of secondary sources is minimized. For these reasons, investigations to date have been concerned principally with the primary sources of dust, and this paper is concerned particularly with one of these—namely, undercutting.

In accord with the foregoing definitions, the sources to which the following opera-

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tions belong have been indicated, basing the decision on the nature of the greater part of the dust in each operation.

Classification of Some Common Sources of Underground Dust

1. Undercutting—majority of dust produced is of primary nature.
2. Drilling—primary source.
3. Blasting or shooting of coal. Information not conclusive as to whether most of the dust is of primary or secondary nature.
4. Loading—secondary source.
5. Underground transfer points—secondary source.
6. Transportation—secondary source.
7. Miscellaneous—inside gobbing, posting and timbering, cleanup, high air velocity, movement of machinery and materials—all secondary sources.

All dust sources produce both primary and secondary dusts. In the foregoing tabulation, an attempt has been made merely to indicate which source of dust is predominant. Though secondary dust sources may not be entirely eliminated by control of primary sources, the importance of these secondary sources can be considerably reduced by proper measures for control of primary dust.

Methods of Dust Sampling and Analysis

Throughout this work, use of standardized methods for determining the concentration of air-borne dust have been considered essential. Of the methods available for sampling such dust, the United States Bureau of Mines' Midget Impinger Method, which has gained general acceptance in this country, has been adopted for these tests. A description of the apparatus and method of use is presented in *Information Circular* 7076. Briefly, the method consists of using a hand-operated vacuum pump to draw dust-laden air through an alcohol-filled impinger tube. In the collection of the dust samples, a standardized procedure was developed for sampling each source of dust, so that comparable results would be obtained. Two methods of analysis for dust concentration were used. One was the microscopic light field as described in the U. S. Bureau of Mines *Information Circular* 7026, which involves an actual numerical count of the number of dust particles contained in an aliquot of the mixture of dust and alcohol, and an arithmetical conversion of this figure to millions of particles per cubic foot. In the hands of a skilled analyst, this method is accurate and reliable, but each dust count requires much time. A careful, patient worker, with good eyes, can make counts on about 12 samples in an 8-hr. day, though the average is between 8 and 10 samples. So that a greater number of samples might be analyzed, a photometric method was also used. In experimental work involving many variables, it is doubtful whether any individual sample is sufficiently representative of a condition to justify the accuracy derivable from a microscopic dust count.

It was realized from the first that, with any photometric method, variations in mean particle size from sample to sample would introduce serious error. For this reason it was used only where: (1) the sample could be collected at a fixed distance from the dust sources, and (2) where the speed of the air currents would not vary unduly.

The principal use of the photometer has been in experimental work on the cutting machine and on a dust survey of cutting and loading conducted over a number of mines. Essentially, the method consists of the insertion of the impinger-collected dust sample, which has been diluted to 50 ml., between a standard light source and a photoelectric cell. Changes in light transmission are recorded on a microammeter connected in series to the photoelectric cell. Any commercial photometer can be used for this purpose.

The photometer must be previously calibrated against microscopic dust counts

on corresponding samples, so that microammeter readings can be converted readily into millions of particles per cubic foot. It is necessary that the instrument be checked at frequent intervals against such dust counts. A drawing of the instrument used is shown in Fig. 1. With a knowledge of the limitations of the method and intelligent observance of sampling rules that will include these limitations, it has been found that this method is capable of an accuracy of plus or minus 10 per cent of the true count. Table 1 shows comparative results obtained with the two methods.

TABLE 1.—*Comparison of Microscopic Light-field Count and Photometric Methods of Dust Analysis*

Microscopic	Photometric	Microscopic	Photometric
Millions of Particles per Cubic Foot			
103	104	84	95
62	62	42	48
37	38	21	21
22	22	11	11
13	15	10	9
6	6	17	20
149	152	143	152
75	72	107	98
37	36	214	212
18	17	34	29
102	104	72	71
102	97	96	97
40	35	212	181
135	127	34	31
65	70	43	48

The figures in Table 1 were derived over a period of six months as a result of weekly checks between the two methods. It should be emphasized that all of these samples were obtained from the cutting operation and were taken in such a manner as to eliminate, as far as possible, variations in mean particle diameter. Strict standardization of the method is essential.

INTERPRETATION OF RESULTS SHOWING DUST CONCENTRATION

The principal problem involved in evaluating dust counts is the wide range of results obtained in a single mine; often the range of dust concentration from a given source in a single section of the mine may be nearly as wide as the range over the mine itself. Table 2 shows dust concentrations that have been determined

TABLE 2.—*Variations in Dust Concentrations Occurring during Undercutting*

Mine	Number of Samples	Dust Concentration, Millions Particles per Cu. Ft.			Ratio of Maximum to Minimum
		Minimum	Maximum	Average	
		FOR SEVERAL MINES			
A	14	83	950	382	11.4
B	32	17	224	77	13.2
C	42	15	215	91	14.3
D	25	79	656	152	8.3
E	13	86	400	203	4.7
F	8	73	600	335	8.2
G	28	48	250	104	5.2
Average for 7 mines					9.3
FOR AN INDIVIDUAL MINE, BY SECTIONS					
Section 1	7	79	191	128	2.4
2	4	82	250	185	3.0
3	5	146	656	155	4.5
4	6	91	175	155	1.9
5	3	130	170	139	1.3
Average for 5 sections in a single mine					2.6

during undercutting for several mines and for sections of one mine.

It is obvious that with such variations between maximum and minimum dust concentrations only an average of a number of samples will properly represent a condition, certainly not one or two samples. This wide variation in dust concentration from a single source makes the problem of obtaining experimental information difficult, particularly if the fundamental reasons for these variations are examined. Some of these factors are listed herewith, but no attempt has been made to place them in the order of their importance. Variations in dust concentration from undercutting may be due to differences in: (1) bed moisture and drainage, (2) ventilation, (3) speed of cutting, (4) condition of cutting bits and bit setting, (5) type of cut,

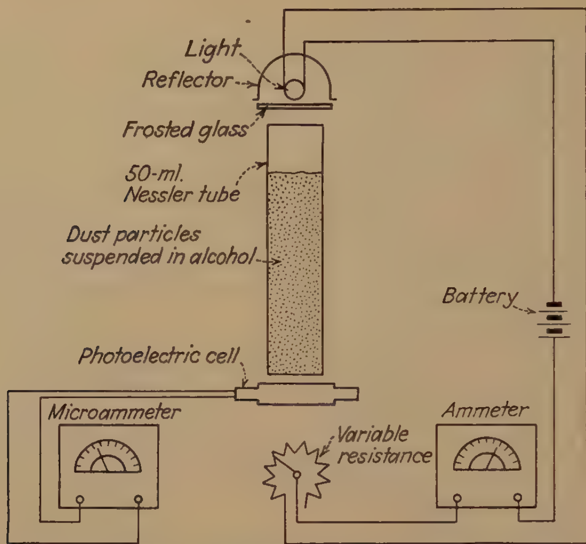

FIG. 1.—DIAGRAM OF PHOTOMETER FOR DETERMINATION OF DUST CONCENTRATION IN IMPINGER-COLLECTED SAMPLES.

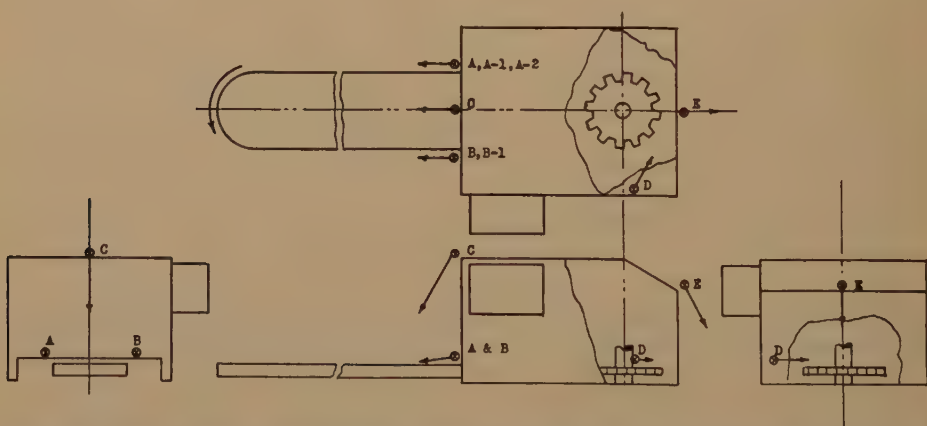

FIG. 2.—VARIOUS SPRAY ARRANGEMENTS TESTED ON SHORTWALL CUTTING MACHINE. ARROWS INDICATE SPRAY DIRECTION.

A. Hollow-cone spray at right-hand lower front spraying at 45° angle.

A-1. Fishtail spray at approximately same position as *A* but directed along cutter chain as it enters kerf.

A-2. Pencil-like jet in same position as *A* but impinging on cutter chain just outside cutter bar.

B. Hollow-type cone spray at lower left-hand front of machine body spraying down at an angle of 45° against stream of cuttings as it passes under body of machine.

B-1. Fishtail spray in same position as *B* and bracketing stream of machine cuttings.

C. Hollow-cone spray at upper front of machine (on center of cutter bar) directed down against cutter bar.

D. Hollow-cone spray near rear tail sprocket.

E. Hollow-cone spray at upper rear of machine body and directed down against cuttings as they are forced from under body of machine.

F. Combination of conditions *A*-2 and *B*-1.

(6) manner in which dust is removed from the cutting machine, (7) presence of clay spars, sulphur balls, etc., (8) quantity of clay or rock strata underlying coal and included in cut, (9) type of mining.

Of these nine conditions, 1 to 8 inclusive vary widely over a single section of a mine and in a single working place various combinations of these conditions will appear as the face advances. Ventilation is particularly important and when the limited means available for quantitative measurement of low-velocity air currents are considered, the difficulty of compensating or correcting for such variation is apparent.

Dust rarely comes from a single dust source. For example, before undercutting is started some dust, principally of a secondary nature, is present in the air. As the cut proceeds, some of the dust that was present in the air before the cut was started settles out and is replaced by dust from the cutting operation proper. The rate at which this secondary dust will be removed depends principally on ventilation, and attempts to apply a correction based on measuring and comparing individual ventilation rates with some average conditions have proved unsatisfactory. To apply at least an approximate correction for the secondary dust, the dust concentration measured before sumping is deducted from the dust measured during sumping and cutting.

The method of sampling and the interpretation of results depend upon the purpose of sampling. In experimental work, it is necessary to standardize on a given set of conditions and limit the sampling to such operations as conform to these conditions, but in conducting a survey all types of conditions must be sampled and much larger variations inevitably occur.

Methods of Allaying Dust from Cutting Machine

A large part of the work to date has related to the dust from cutting-machine operation. A number of nozzle locations and types of sprays have been tried. Each arrangement was sampled for dust concentration over a long enough period of time to afford a reasonably accurate basis of comparison. Fig. 2 is a diagram showing the various locations and types of sprays used in these tests and also serves as an identification key for Table 3, which shows the comparative effectiveness in suppressing dust of sprays located in the positions thus described.

Table 3.—*Effectiveness of Various Spray Arrangements on Shortwall Cutting Machine*

Average cutting speed, 1.4 ft. per min.
Cuttings produced (avg.), 275 lb. per min.
Spray water, 0.45 gal. per min. per spray nozzle
Pressure, 35 lb. per sq. in.
Nozzle orifice, 0.065-in. diameter

Spray Arrangement (See Fig. 2)	Dust Concentration, Million Particles per Cu. Ft.: Unsprayed Undercut[a]	Dust Concentration, Million Particles per Cu. Ft.: Sprayed Undercut	Water Treatment, Gal. per Min.	Reduction in Dust, Per Cent
A	188	18	0.45	90
A-1	188	13	0.45	93
A-2	188	30	0.45	84
B	188	33	0.45	83
B-1	188	26	0.45	86
C	188	71	0.45	62
D	188	89	0.45	53
E	188	78	0.45	59
F	188	8	0.90	96

[a] Represents average dust concentration during sump and cut minus average dust during 2.5-min. period just before sumping started. The unsprayed dust concentration (188) is an average of samples of unsprayed undercuts taken throughout the series of tests.

The figures in Table 3 are based on a minimum of 10 cuts for each spray arrangement. Even with this number of cuts, it is believed that one arrangement cannot be considered as definitely superior to another unless the difference in dust concentration is 10 per cent less with the first than with the second. For this reason, it was concluded that all of these arrangements except C, D and E would probably give satisfactory performance and that the final selection must depend on other considerations. Arrangement F, which is

a combination of A-2 and B-1, was selected as the standard spray installation on cutting machines for the following reasons:

1. The application of spray water from two points allows a certain safety factor in that if one spray clogs the dust will be greatly reduced by the other.

2. In these positions the sprays are accessible for inspection and repair.

3. In such position they can be protected against damage.

4. The operation of sprays at these points is not affected by the manner in which the cutting machine is operated or the cuttings removed.

Of the various types of nozzle used, the fishtail was selected for the left side and a single jet for the right. With this combination and location extended tests in a number of sections of three mines have shown that a dust concentration of 200 to 300 millions of particles per cubic foot (unsprayed) can be reduced by 80 to 90 per cent by using a total from both sprays of 0.45 gal. per min. at a pressure of 35 lb. per sq. in. This was accomplished by reducing the orifice diameter to 0.040 in. When more than 0.45 gal. per min. was used, it did not reduce the quantity of dust enough to justify the increased quantity of water.

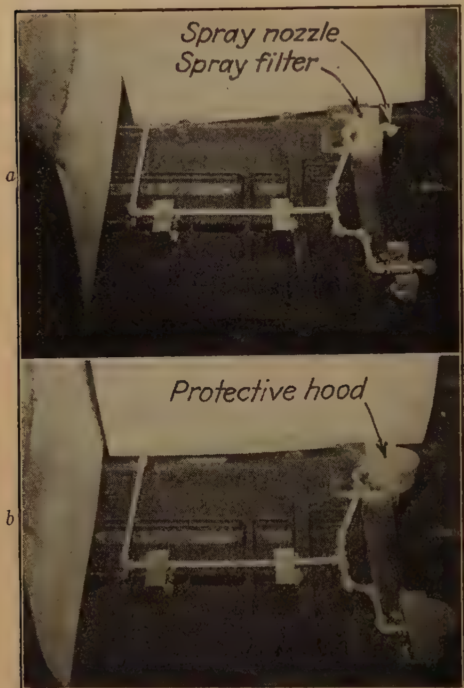

FIG. 3.

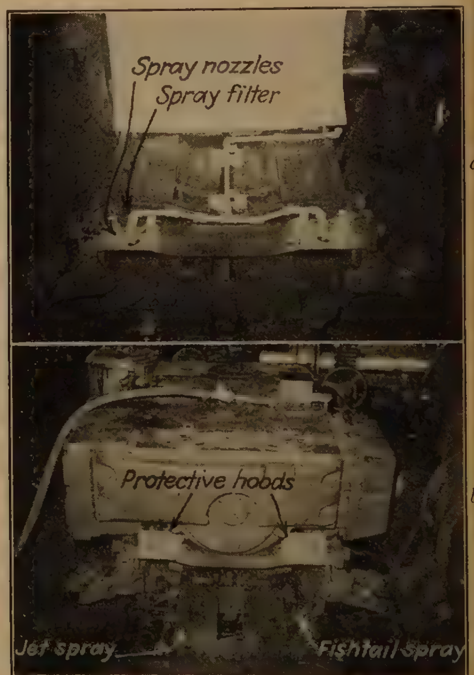

FIG. 4.

FIG. 3.—SIDE VIEWS OF SPRAY ARRANGEMENT ON SHORTWALL CUTTING MACHINE.
a. Piping, spray filters and spray nozzles (C. M. motor removed).
b. Piping and hoods protecting spray nozzles (C. M. motor removed).

FIG. 4.—FRONT VIEWS OF SPRAY ARRANGEMENT ON SHORTWALL CUTTING MACHINE.
a. Piping, spray filters and spray nozzles (C. M. motor removed).
b. Protective hoods and sprays directed on cutter bar (C. M. motor in place).

SPRAY ASSEMBLY ON CUTTING MACHINE

To ensure uniformity in installations and to simplify the maintenance of the spray arrangement selected for the cutting

machine, a set of standard, interchangeable parts was designed for each type of shortwall cutting machine. Figs. 3 and 4 show photographs of the spray and piping assembly on one of the shortwall undercutters. A year of use has demonstrated the practicability and durability of this installation. Attention is called to the use of a main-line filter and an individual filter at each spray nozzle. These filters are packed with cellulose sponge and steel wool. Tests of these materials have proved them superior to a number of commercially available filters for this purpose. Proper filtration and protection of both sprays and piping are essential if satisfactory performance is to be maintained. Too much emphasis cannot be placed on these two points—filtration and protection of spray nozzles—as experience shows that often spray systems receive little attention and, unless the design is nearly foolproof, the spray will not give continuous and efficient performance.

Pressure Required for Shortwall Cutting-machine Sprays

In various publications it has been stated that minimum spray pressures of from 75 to 125 lb. per sq. in. are required to effect proper reduction of dust. It appears to be a general opinion that only by atomizing the spray water can the water be applied efficiently so as to allay dust produced by shortwall cutting machines. Recent publications have reported dust reductions up to 90 per cent, with the claim that this reduction was effected without wetting the coal but by an attack on the dust in suspension. Air-conditioning experience has shown the difficulty of obtaining any large percentage of reduction on a particle-count basis by ingenious combinations of ultrafine mists. If reduction of the fine dust by mists is difficult in air-conditioning equipment, it is hardly to be expected that a spray or mist alone applied to a cutting machine will remove any large proportion of the air-borne dust. The conditions stated in most publications indicate that there is little evidence to show whether the dust-forming fines were wetted before they were dispersed as air-borne dust or after they were suspended in the air. It is difficult to see how the sprays on a cutting machine could be placed in the usual positions without a large part of the water moistening much of the fine dust before any of it rises into the air. It cannot be stated that the use of fine mists and possibly wetting agents will not facilitate the removal of any considerable percentage of the air-borne dust from the air, but our experiments positively indicate that, with the shortwall cutting machine, it has been unnecessary to use sufficient pressure to produce such a mist. In the course of this investigation, the method of water application of the shortwall cutting machine has produced an 80 to 90 per cent dust reduction with quantities of water so small (0.2 gal. per min.) that those high pressures are impractical, owing to necessity for small spray-nozzle orifices. Even with proper filtration, orifices smaller than 0.030 in. are impractical. This small quantity (0.2 gal. per min.) cannot be delivered at 75 to 125 lb. per sq. in. pressure unless the orifice is reduced to such a small opening that available filters will not prevent it from clogging. The rather coarse sprays, produced at 35 lb. per sq. in., and the fact that half of the water is not sprayed at all, but applied as a jet, certainly indicate that with the shortwall cutting machine these high pressures are not essential for satisfactory dust reduction.

Use of Wetting Agents and Their Effect on Dust Reduction

At many mines the cost of supplying water to the face and the necessity for lower moisture in the coal has indicated that the use of wetting agents to decrease the quantity of water needed to allay dust might have advantages. Twenty-five rea-

gents recommended by the manufacturers for their wetting properties were tested in the laboratory, using two types of tests. Two of the most promising wetting agents (A and B in Table 4) were selected from a consideration of their performance and cost. The third reagent selected was a low-viscosity oil, which forms a stable oil-in-water-emulsion at dilute concentrations. It appeared to have the additional property of preventing stud bolts holding the cutting bits from being corroded by the sprays. The two wetting agents A and B

TABLE 4.—*Effect of Wetting Agents and Emulsion-forming Oil on Dust Reduction with Shortwall Cutting Machines*

Spray pressure, 35 lb. per in. Nozzle orifice, 0.040 in. Spray water, total 0.2 gal. per min. from two nozzles, Spray Arrangement F (Fig. 2)

Reagent	Reagent Concentration, Per Cent	Reagent Cost per 100 Gal.	Reduction in Dust, Per Cent
Water			26
Wetting agent A	0.17	$0.21	54
Wetting agent B	0.2	0.21	60
Wetting agent C	0.1	0.30	71
Emulsion-forming oil	0.4	0.21	77
Emulsion-forming oil	0.2	0.125	60

were tested on the cutting machines over an extended period of time. Wetting agent C, Table 4, showed promise but was not tested underground until a later date.

With water alone or with water plus either of the two wetting agents, the corrosion of the threads of stud bolts holding the cutting bits in position increased so rapidly that after a few weeks of spraying it was necessary to shop each machine to loosen these bolts. Even after operation for several days, sufficient corrosion occurred to cause delays in changing bits. No unusual corrosion was evident on any other part of the machine. The low-viscosity oil in the spray water prevented the binding of the stud bolts and also aided in the reduction of dust.

Table 4 summarizes the results on the effect of wetting agents and emulsion-forming oil on dust reduction with shortwall cutting machines. The concentrations of wetting agents and the emulsion-forming-oil were adjusted so as to give the same cost per 100 gal. of water.

From the data in Table 4, it is evident that, on the shortwall cutting machine, the emulsion-forming-oil, when used in concentrations of equivalent cost per 100 gal. of spray water gave the best dust reduction. Also, the use of the material for nearly a year at one mine has eliminated entirely the previous trouble with the stud bolts. This advantage gained by the use of the emulsion-forming oil does not alter the fact that wetting agents are extremely helpful in dust reduction.

PROPERTIES OF WETTING AGENTS

The properties of the dilute oil emulsion and a wetting-agent solution are entirely different. In the oil emulsion used, the emulsifying agent has lowered the interfacial tension between oil and water and tends to concentrate at the oil-water interface. Because of this tendency to concentrate at that contact, there appears to be little reduction of the interfacial tension between the emulsion and coal; therefore, there is little difference between the spreading action of the oil emulsion on coal and that of plain water.

On the other hand, the use of wetting agents to reduce the coal-water interfacial tension as shown in Table 4 increases dust reduction. Therefore, it appears that there are two entirely different effects that may cause greater dust reduction than plain water. Wetting agents reduce dust by increasing the wettability of the coal whereas with the dilute oil emulsion the dust reduction is attributable to the presence of a small quantity of oil.

In the action of a shortwall cutting machine, two conditions are present that have a considerable effect on dust reduction and the use of wetting agents:

1. An appreciable time interval between contact of bit with coal which produces fines and ejection of cuttings from rear of machine.

2. The turbulence and packing which occurs as the cuttings are produced and as they travel along the chain, and pass under the machine.

Even water alone will have an opportunity to be distributed over the surface of the machine cuttings by "surface smear" if water is introduced at a point to allow the above factors to operate. The dust reduction effected by the dilute oil emulsion is attributed to the distribution of the oil over a wide surface with the aid of this time factor and turbulence.

Assuming that two distinct and separate factors operate with oil emulsion and with wetting agents, tests are being conducted on the use of blends of oil-forming emulsion with sufficient wetting agent to saturate the oil-water interface and still reduce the coal-water interfacial tension. Information is incomplete on this phase, but this feature appears to have possibilities.

Tests of the dilute oil emulsion on conveyor discharge points have not shown any marked benefit over water whereas all three of the wetting agents tested have shown greater dust reduction than water alone. In these cases, the spray water strikes the coal just a fraction of a second before the dust is dispersed and the time factor, turbulence and pack'ng that accompany the cutting-machine operation are absent.

Effect on Secondary Sources of Primary Dust Reduction

According to the definitions of primary and secondary dust sources, it follows that any reasonably efficient method of dust suppression applied to a primary source will be reflected in a reduction of dust from secondary sources. For example, if the dust from cutting and drilling is suppressed by sprays, the dust arising from loading, cleanup, timbering, and movement of men or machines will be less than if no sprays were used while cutting and drilling. Of course, this is true only if the method of dust control has so fixed the dust-forming fines that they cannot be subsequently dispersed. It is better to bind the finest dust than to allow it to become dispersed and then attempt to remove it from the air at the primary source.

To determine the effect of undercut spraying on secondary dust, samples were taken of this dust over a period of time in which 80 per cent of all the cuts had been sprayed. The results shown in Table 5 are ample evidence of the point previously stated; namely, that if the primary dust sources are adequately controlled, the importance of secondary sources is minimized. These samples were taken while the rooms were being cleaned up prior to undercutting and after the second week of spraying the dust dispersed during this clean-up operation had been reduced by 66 per cent. Furthermore, the dust concentrations are low.

Table 5.—*Influence of Spraying Undercut on Secondary Dust Produced in Preparation of Room for Next Cut*

Spray water, 0.45 gal. water per min.
Spray pressure, 35 lb. per sq. in.
Nozzle arrangement F (Fig. 2)
Orifice diameter, 0.040 inches

Week	Number Cuts Made	Number Cuts Sprayed	Number of Samples	Dust Concentration before Cut, Millions Particles per Cu. Ft.
First	14	3	14	38
Second	19	16	16	40
Third	17	11	17	14
Fourth	13	13	13	13

Table 5 shows what effect dust-control measures applied to a primary source have on a secondary source. To show this effect in more practical terms, an extended test was conducted over the entire operating section of a mine, which was developed by

driving two butt entries on 350-ft. centers and to a depth of 2150 ft. off the main entry. One of these entries is used as an aircourse and the other for a gathering belt conveyor. Twenty-four rooms, each 18 ft. wide, are turned at right angles from this belt entry and are driven to a distance of 300 ft. The panel is extracted on retreat. As many as nine places are worked at a time, eight being rooms or room pillars and the ninth the chain pillars. Shaking conveyors transport the coal from the face to a gathering belt conveyor on the butt entry. From the gathering belt, the coal is loaded into mine cars as they pass a discharge point on the main heading. In this section, the principal secondary sources of dust are:

1. Nine (9) transfer points from shaking conveyors to gathering belt conveyor.
2. Main loading head at junction of gathering belt and main heading.
3. Cleanup along belt line and around loading head.
4. Movement of men, supplies and machinery.
5. Hand loading of coal into shaking conveyors at working face.

The principal primary dust sources are undercutting and drilling.

Tests in which no sprays were used on the cutting machines were run in this section for one month and tests in which every cutting machine was equipped with sprays were run for one month. During both periods, a spray system was operated at the main loading head. Rock-dust samples were taken weekly throughout the belt area and around the loading head in a standardized manner, at intervals of 20 ft., and a careful record was kept of rock dust applied during both periods. From the cumulative average rock dust applied per hundred linear feet and the weekly average of noncombustible in the rock-dust samples, the average quantity of coal dust deposited per hundred linear feet of entry per shift was computed. Table 6 summarizes the data from this test.

Much more dust reduction, as measured by the rock-dust method, was obtained in the loading-head zone than the samples of air-borne dust taken with the Midget Impinger showed, because the air-borne dust was sampled down the entry about 15 ft. from the loading head (in the direction of air movement) whereas the rock-dust samples were obtained at points in a zone lying between the loading head and a distance of 160 ft. therefrom and in the same direction.

TABLE 6.—*Effect of Spraying Undercuts on Secondary Dust in Section, Using Data Derived from Rock Dust Applied and Incombustible in Rock Dust*

Sprays[a]	Coal Dust Deposited per Shift per 100 Lin. Ft. of Heading, Lb.		Air-borne Dust Collected 15 Ft. from Loading Head in Direction of Air, Millions of Particles per Cu. Ft.
	Belt Area	Loading Head Area	
No sprays on cutting machine	7.6	28.9	177
Sprays on cutting machine	3.8	8.2	78
Dust reduction, per cent	50	72	56

[a] Loading-head sprays.

Because reliable statistical data are lacking on the collection and analysis of rock-dust samples, this method may be open to some criticism. However, as identical methods were used in each part of the test, the data for the two conditions are believed to be comparable. Percentage reduction in dust deposited per shift per 100 lin. ft. of heading appears to be a valid basis of comparison, even though the actual quantity of dust deposited may be higher or lower than the figures quoted.

METHOD OF WATER DISTRIBUTION

From an operating viewpoint, the methods used for supplying water to the working face at sufficient pressure to meet

spray requirements is of fundamental importance, but it cannot be considered in itself without a study of the whole picture. The degree of dust reduction to be effected, or the maximum permissible dustiness at a single point, must be considered, for this will determine in part the number of sources to be controlled and the quantity of water required. The degree of effectiveness that will be demanded of a dust program does not lie within the province of the industry to determine, and until this information is supplied any intelligent long-term policy is impossible. Thus, the only possible approach is to try a number of ways for supplying water to the face. During this investigation, all the methods listed below have been considered and, with one exception, tried.

1. Use of pipe throughout the mine with water supply from a central source.

2. Use of pipe in working sections with water supplied from conveniently located dams or sumps, where the dams or sumps are supplied by: (*a*) track-mounted tank car; (*b*) drill hole tapping surface stream; (*c*) connection to mine-drainage system.

3. Use of track-mounted tank car holding about 1000 gal., which may be switched off the main track in a working section with water distributed to face by pipes.

4. Use of track-mounted tank car that is hauled with the cutting or loading machine.

5. Use of tank built as an integral part of the cutting-machine truck.

6. Use of a small hand portable tank holding 5 to 5½ gal. of water charged into tank after an initial charge of about 25 lb. air pressure.

None of these methods can be dismissed without some consideration because, dependent on the distance over which the dust must be allayed, the number of sources of dust to be controlled, the variations in mining development and practice, any of these methods might prove superior to the others from an operating cost and investment viewpoint.

Though, as yet, it has been impossible to determine rigid rules for the selection of the best method of water distribution, all the methods appear to embody certain advantages. Use of pipe from a central water source usually involves a large investment, and experience has shown that in working sections pipe has a relatively short life. Where mining development and extraction are extremely concentrated, this method unquestionably must be considered, particularly where a complete program of dust suppression is required. In mines where the operations are scattered over a wide area and where the program required is of a limited nature, the applicability of this method is doubtful. Where a complete program of dust suppression is required and the working sections are widely scattered, the use of a pipe in working sections only with any of the three methods of water supply noted above in 2*a*, 2*b*, 2*c* or 3, will be applicable. In certain methods of mining, especially where the rate of advance and retreat is extremely rapid, a piping system is being continuously reconstructed, with resulting operating difficulties. When a track-mounted tank is used to accompany cutting or loading equipment, in a mechanized mine, tramming and switching become difficult, therefore in many places use of such equipment is impracticable. A cutting or loading machine was never intended to be used as a motor and, with steep grades, transportation of the tank is difficult or impossible.

Where the problem of dust suppression involves only the application of sprays to the cutting machine, the use of a tank built as an integral part of the cutting-machine truck is to be recommended. This takes care only of the cutting machine and the expenditure is large, seeing that only one dust source is attacked. With this system, a distribution of water to the cutting-machine tank is still required. If there is some question as to how far a dust-suppression program is to extend, it

is probably advisable to consider other methods of water supply, so that additional sources of dust may be sprayed.

In water-supply methods 1 to 5 inclusive, two important factors are involved: (*a*) the superimposing of a new system of supply on a system that already exists, for handling timber, rock dust, etc.; (*b*) the addition of much underground equipment, which requires maintenance and supervision.

With the various methods already discussed, control and supervision of maintenance, motors, pumps, piping and supply

FIG. 5.—HAND PORTABLE SPRAY TANK.
Showing hose connection, valve and tee-handle wrench for valve.

of water to the face becomes a difficult problem, especially where mines are operated on a three-shift basis.

For these reasons, use of hand portable tanks for controlling primary dust at the face may have merit. Each tank (Fig. 5) consists of a 5½-ft. length of a light-weight boiler tubing with tightly welded end plates, fitted with a substantial valve. These tanks are filled and charged in batteries of 5 or 10 tanks at a time. In this system, the only mechanical equipment involved is the tank-charging mechanism, which can be located outside or at any convenient location. The tanks will maintain their charge for a week or more.

Though with this system distribution of water still is necessary, the method is essentially the same as that used in the distribution of Cardox shells and other supplies. These tanks are suited for moistening dust at any of the face operation, such as cutting, drilling and loading and will involve a relatively small capital investment. They cannot be applied to fixed sources of dust, such as loading heads and underground transfer points. Such units merit a separate installation for dust suppression.

ACKNOWLEDGMENT

By the very nature of this problem, individual efforts alone are of limited effectiveness. Recognizing this, the authors wish to emphasize that such progress as has been made by the Pittsburgh Coal Co. in dealing with this problem has been due entirely to the cooperative efforts of so many individuals in the Company, that a tabulation of their individual names is impossible. We are particularly indebted to Messrs. J. B. Morrow, H. C. Rose and H. F. Hebley, for permission to publish these data.

REFERENCES

1. A. M. Bryan and J. Smellie: The Estimation of Dust in Mine Atmospheres. *Jnl.* Roy. Tech. Coll. Glasgow (Scotland) **3**, 4 (1936).
2. A. M. Bryan: Notes on Some Dust Problems in Coal Mines. Mine Inst. Scotland, *Trans.* (Apr. 27, 1938) **58**, Pt. 2 and 3, 36–8.
3. A. L. Davies: Abatement of Nuisance of Air-borne Coal Dust in Intensive Conveyor Mining. S. Wales Inst. Engr. Proc. *U*-52 (1936) 293–304.
4. R. H. Flinn et al.: Soft Coal Miner's Health and Working Environment. U. S. Pub. Health Service, *Bull.* 270 (1941). 118 pp.
5. J. J. Forbes and A. H. Emery: Sources of Dust in Coal Mines. U. S. Bur. Mines *R.I.* 2793 (1927) 17 pp. *Trans.* A.I.M.E. (1927) **75**, 645–664.
6. T. C. Futers: Mine Dust Settling Apparatus. *Coll. Guard.* (May 31, 1935) **150**, 979–980.
7. I. Graham: Means for Reduction of Dust with Special Reference to Mechanized Mining. *Coll. Guard.* (Apr. 14, 1939) **158**, 677–678.
8. J. I. Graham, D. G. Skinner and W. H. Walton: The Reduction of Dust in the Vicinity of Underground Loading Points. *Trans.* Inst. Min. Engrs. (1937–38) **94**.
9. J. I. Graham, D. G. Skinner and W. H. Walton: The Production of Dust in Hand Headings and Means for its Suppression. *Trans.* Inst. Min. Engrs. (1938–39) **97**.
10. J. H. Griffiths and W. L. Gyles: Sprays for Dust Suppression: *Iron and Coal Tr. Rev.* (Oct. 11, 1940) **141**, 362; *Coll. Guard.* (Oct. 11, 1940) **161**, 366–368; *idem* (Oct. 18, 1940) **161**, 389, 391–392.
11. D. Harrington and J. J. Forbes: Forwarding Health and Safety in Coal Mining with Watering Methods. U. S. Bur. Mines *I.C.* 7041 (Dec. 1938).

12. D. Harrington: Formation, Treatment and Collection of Dust in Mines. U. S. Bur. Mines *I.C.* 7089.
13. I. Hartman and H. P. Greenwald: Use of Wetting Agents for Allaying Coal Dust in Mines. U. S. Bur. Mines *I.C.* 7131.
14. J. L. Hay and F. V. Tideswell: The Treatment of Dust on Traveling Roads. *Trans.* Inst. Min. Engrs. **90** (1935–36) 213–223.
15. C. W. H. Holmes: Coal Dust. *Coll. Guard.* (Apr. 1939).
16. E. T. John: Method of Clearing Gas and Allaying Dust in Track of Coal Cutter. *Trans.* Inst. Min. Engrs. (Aug. 1940) **99**, pt. 5, 226–232.
17. C. W. Owings: Methods of Allaying Dust in Underground Mining Operation. U. S. Bur. Mines *R.I.* 3631.
18. C. W. Owings: Some Preliminary Data on Methods of Controlling the Dust Hazards in Mechanical Mining. U. S. Bur. Mines *I.C.* 7151 (1941).
19. A. C. Peterson: What One Anthracite Company is Doing to Solve Dust Problems. *Coal Age* (Nov. 1941) **46**, 63.
20. H. Price: "Perminal W" and Calcium Chloride Treatment of Haulage Roads. Inst. Min. Engrs. *Trans.* (June 1937) **93**, pt. 3, 252–260.
21. J. A. Saxe: Allaying Dust at Island Creek Coal Company Mines. *Jnl.* Min. Congress, Vol. 28, Jan. 1942, pp. 15–17.
22. R. J. Shaw: Suppression of Coal Dust. Iron and Coal Tr. Rev. (Jan. 5, 1940) **140**. Discussion: *idem* (May 10 and 17, 1942) **140**.
23. J. B. Sproston: Some Methods of Dealing with Underground Coal Dust. *Trans.* Inst. Min. Engrs. (March 1936) **90**, pt. 6, 350–355.
24. F. V. Tideswell and R. V. Wheeler: Dust Laying on Mine Roadways. *Trans.* Inst. Min. Engrs. (1933–34) **87**, 1–8.
25. J. Walker: Suppression of Dust in Mines. *Coll. Guard.* (Feb. 2, 1940) **160**, 175–177. Discussion: *idem* (Apr. 19, 1940) **160**, 575.
26. J. Walker, S. Whitefield and W. R. Evans: *Trans.* Inst. Min. Engrs. (1940) **99**, pt. I, 8–14.

DISCUSSION

(*David R. Mitchell presiding*)

H. P. Greenwald,* Pittsburgh, Pa.—The authors of this paper are to be congratulated for presenting clearly and concisely the results of an investigation that evidently consumed much time and effort. They have emphasized the need for a definition of what might be termed "permissible maximum air dustiness." Note that this is actual dustiness at a given instant and not potential dustiness that would prevail if all quiescent dust in the vicinity were brought into suspension. Consequently, the principal factors entering into determination of this permissible maximum are the hazard to health and visibility. Health authorities are conducting investigations that will lead ultimately to clarification of this problem and probably to definition of the permissible maximum that the authors desire in the not too distant future.

Discussion of methods of making dust counts brings out the difficulty of compromising two opposing factors—accuracy and speed. A final solution of this problem seems unlikely until the permissible maximum and tolerances in its determination are fixed, but accumulation of a mass of data on the accuracy of rapid methods is greatly to be desired.

The authors' data are further support of the old saying, "kill the dust where it is made." Reduction of primary dust automatically reduces secondary dust. It is to be hoped that further data along this line will be forthcoming. Our knowledge is still too incomplete.

One point not elaborated in the paper is the fact that this investigation was conducted by a company organization with a minimum of aid from outside organizations. Bituminous-coal mining is too large an industry to have its problems solved by outsiders if it expects to retain full control of its affairs. There must be within the industry knowledge, ability, and willingness to face the facts and to solve problems logically. Certainly, the authors should be encouraged to continue their work, and the industry would benefit if many others could find it possible to supplement their efforts.

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C. W. Owings,* College Park, Md.—The problem of control of dust in coal mines is complex and requires considerable research to determine the most effective method of allaying dust under the many different conditions found today in coal mines of the United States. The Pittsburgh Coal Co. has undertaken a comprehensive study of the subject and the paper presented by Davis and Gardner testifies to the thoroughness with which the investigation is being conducted. Their findings are of definite value to operators of bituminous coal mines.

The authors discount the effectiveness of fine sprays with the requisite pressures, often stated to be 75 to 125 lb. per sq. in., and cite air-conditioning research as an example. The two problems are not necessarily comparable. The question of the proper pressure for maximum benefit from sprays has been investigated by the Bureau of Mines under actual mining conditions. The type of equipment forming dust determines to a large extent the fineness of a spray and the pressure necessary to obtain this spray. With shortwall mining machines, dust counts of 20 million particles per cubic

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foot of air, or under, have been obtained, with a pressure of 20 lb. per sq. in. On the other hand, satisfactory results have not been obtained with center-cutting and top-cutting machine pressures of 40 and 45 lb. per sq. in. Allaying coal dust with water and allaying it with water containing a wetting agent have two different results, primarily because wetting agents tend to reduce greatly the interfacial tension between the liquid and the dust. For this reason it has been found that the finer sprays, which allow application of considerably less liquid, are more effective where wetting agents are used. Tests conducted under widely differing physical conditions and in a number of different coal fields have shown that for cutting, particularly with track-mounted machines, and more especially those cutting in the center and top of the coal bed and with mobile loading machines, a fine spray with pressures of approximately 75 lb. per sq. in. are relatively effective compared with other known methods. Recent commercial investigations have also demonstrated the fact that water containing an active wetting ingredient, when introduced into air-conditioning systems as a fine mist, reduces the dust in the air to a much greater degree than is possible with water in any form.

The results obtained by the Pittsburgh Coal Co. with low pressures show clearly, however, that it is possible to obtain a large reduction in dust at low pressures when undercutting with shortwall mining machines.

Application of Chemistry in Combatting Anthracite Mine Fires

By G. S. Scott,* Member A.I.M.E., and G. W. Jones*

(Easton Meeting, October 1941)

Economic waste caused by mine fires may become considerable,[15] especially if a fire is allowed to spread or temporarily to get beyond control. It is important, therefore, to act promptly whenever a fire is suspected and to make every action count. To contribute toward its extinction effectively it is necessary to know at the earliest possible moment, and continuously thereafter, what is happening at the seat of the fire, which may be in a remote or inaccessible area, and also to recognize what may be expected as a result of any proposed combative measure.

In combatting a fire the mining engineer and the chemist should cooperate, first deciding upon the course of action to be taken and then drawing upon the mining engineer's intimate knowledge of the physical structure of the fire area, the cost and time required to carry out proposed operations, and the probable efficiency with which these operations could be performed, and upon the chemist's knowledge for the results to be expected therefrom.

After the initial steps to be taken have been decided, it devolves upon the mining engineer to see that they are carried out and upon the chemist to evaluate the results and to keep the mining engineer continuously informed in regard to a variety of matters—such as the composition and toxic or asphyxiating properties of the atmosphere in which the men may be working, whether the fire is increasing or decreasing in intensity and its probable temperature, the quantity of air leakage and the explosion hazards of the fire area.

The purpose of this paper is to discuss what the chemist can and cannot contribute as a member of the fire-fighting personnel.

The technical literature contains several hundred papers relating directly and indirectly to mine fires, and this fact alone is testimony to the complex nature of the phenomena involved in the birth, life, and final demise of a fire. A fire may be stopped (1) by preventing oxygen from reaching the coal or (2) by removing the conditions necessary for the rapid reaction of coal with oxygen; i.e., decreasing the temperature. All methods of prevention or control necessarily must be based upon one or the other, or both, of these two principles.

The chemistry of a fire may be understood readily if the low-temperature oxidation of coal is considered as a starting point; then oxidation at higher temperatures (or combustion); and with these, the composition of the solid and gaseous products resulting from the oxidation or from the effects of the heat liberated. This discussion refers particularly to Pennsylvania anthracite, although much of the material applies equally well to coals of lower rank.

Oxidation of Coal

Oxidation of coal fundamentally is confined to the surface; that is, the rate at which oxygen penetrates into the interior of a solid block of coal is so slow that it may be ignored.[6,16] Oxidation starts when a fresh surface is exposed, and the rate is

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[15] References are at the end of the paper.

proportional to the amount of this surface. There is some evidence[32] that immediately upon the creation of a fresh surface there is a slight but very rapid union with oxygen, strongly resembling physical adsorption. Following this are the regular chemical reactions between oxygen and the coal substance itself.

In spite of the vigorous and persistent attempts of many investigators to discover the fundamental chemical compounds of which coal is composed, little dependable information has been obtained. The fundamental nature of the oxidation processes therefore must also remain somewhat obscure, since knowledge of the former is necessary for understanding of the latter. The following picture therefore is suggested as a working hypothesis that fits the known facts, and the purpose in presenting this is not to suggest any theory but solely to facilitate understanding of the oxidation processes.

There is some evidence that coal is a "hydrocarbon";[13] i.e., that the hydrogen and carbon are in combination and not in the elementary state. Therefore it is suggested that the carbonaceous matter of coal consists essentially of a continuous series of compounds in which there is a regular variation of the hydrogen-carbon ratio. It is a known fact that the ratio of hydrogen to carbon in coal is decreased by oxidation, so it is suggested that the compounds of higher hydrogen-carbon ratios are more readily attacked by oxygen than those of lower ratios. This is also in agreement with the known fact that lower-rank coals displaying higher hydrogen-carbon ratios than the higher-rank coals are more readily oxidized than the latter.

At temperatures below about 400° to 500°C. the oxygen consumed in the oxidation processes forms carbon dioxide, carbon monoxide, water, and a solid "coal-oxygen complex."[19,28] It is presumed that the presence of this complex retards the oxidation processes, as, other conditions being constant, the rate of oxidation decreases with increase in the amount of complex present and as removal of this complex by distillation increases the activity of the coal toward oxygen.[2] Further, the solid "coal-oxygen complex" does not appear at temperatures above about 500°C., and above this temperature there appears to be no retarding of the oxidation rate except what may be ascribed to changes in surface area exposed.

It has been found experimentally that the major factors that influence the oxidation rate of a given anthracite at low temperatures are: (1) concentration of oxygen in contact with the coal,[23] (2) temperature,[26] (3) extent of previous oxidation,[25] and (4) surface area of the coal.[22]

Factors Involved in Heating and Cooling

Starting at, say, 30°C., fresh coal in contact with oxygen will oxidize with liberation of heat.[20] This heat will cause the temperature of both coal and air to rise. The temperature difference thus established creates a chimney effect within the pile of coal, which causes air to circulate through it. This circulation not only supplies oxygen for further oxidation but also carries away heat. As the temperature increases, the oxidation rate tends to increase; but during the oxidation processes the coal-oxygen complex is formed, and this tends to reduce the rate.

In spontaneous heating the net heat available for temperature rise is zero when the heat produced is equal to the heat lost. This point is reached when the coal-oxygen complex decreases the rate as much as the increase in temperature increases it. After reaching this point the coal must cool, because of further increase of the complex and consequent retardation of the rate. For simple cases the time-temperature curve for a body of exposed coal can be calculated, and the maximum temperature it may be expected to attain can be obtained there-

from, as well as the total quantity of oxygen the coal would be expected to consume for any period during this exposure.

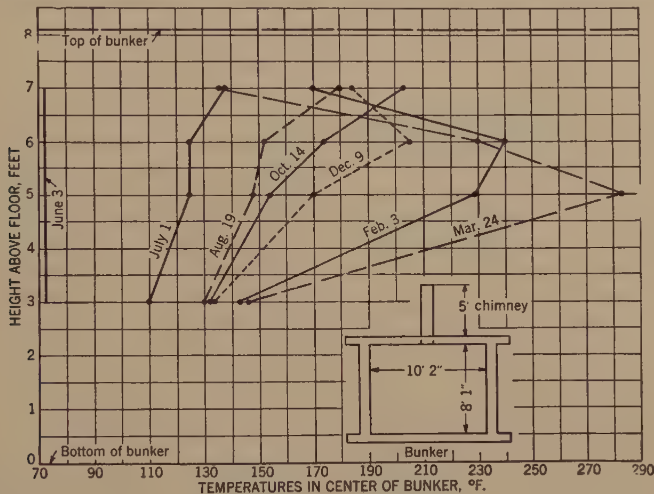

FIG. 1.—SPONTANEOUS FIRING OF COAL FROM WARWICKSHIRE SEAMS. (GRAHAM AND RAYBOULD.[7])
Size of coal: 80 per cent through 4 mesh; 13 per cent through 100 mesh.

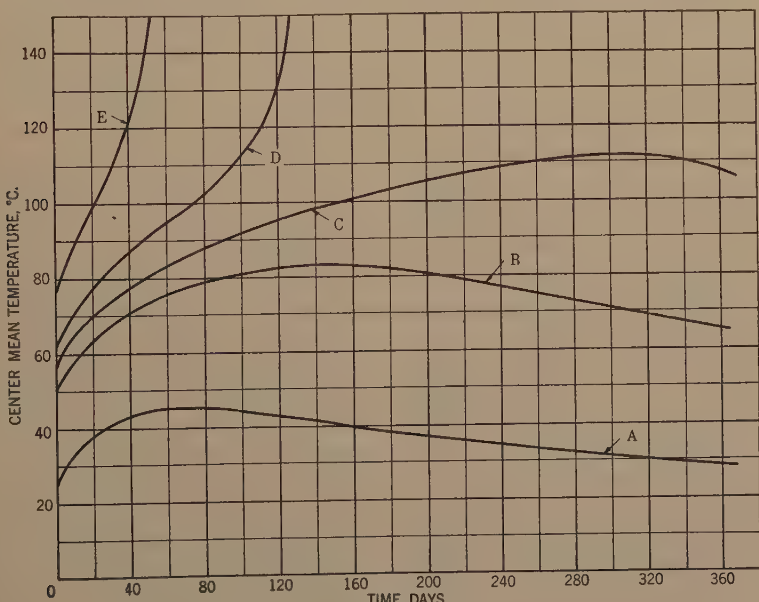

FIG. 2.—TIME-TEMPERATURE CURVES BY CALCULATION FOR A PILE OF COAL.

The latter figure may prove useful, as will be seen later.

During the initial heating period the heat liberated is in part carried along the path of air circulation, so that there is a cumulative effect, the highest temperature being at the point where the air leaves the body of coal. As the temperature rises the oxygen in the emerging air becomes less and less, so that oxidation near the point of emergence is retarded; the most active

oxidation then takes place in a region below the point of emergence. Continuation of this process causes the seat of most active oxidation (or hot spot) to travel toward the source of oxygen supply. This is well demonstrated by the data of Graham and Raybould,[7] some of which are shown in Fig. 1. (The air enters at floor level, here.)

Fig. 2 illustrates plots of several time-temperature curves calculated for a large bank of a certain screen-sized anthracite. The temperature of the surrounding air was assumed constant at 25°, 50°, 55°, 60°, and 75°C., respectively. Curves *A*, *B* and *C* show temperature rises to maxima followed by cooling. Curves *D* and *E* show spontaneous heating that will result in ignition. The ordinate scale, of course, will depend upon the size of bank, the particle size and freshness of the coal, and its specific oxidation rate.

Curves of this type have been obtained in the laboratory by Winmill,[33] Jäppelt,[8] and Agde and Gölz,[1] and of the type of curves *D* and *E* by Davis and Reynolds.[3]

In field measurements the curves would be expected (1) to depart somewhat from the smoothness shown because of daily variations in wind velocity, atmospheric temperature, and pressure, and (2) to show a somewhat slower initial rise because time is required to build a bank and because the coal is building up the oxidation-resisting complex during this time. When these facts are considered the curves of Fig. 2 will be found to agree well in shape with the actual measurements of coal-pile temperatures made by Fayol[4] and others.

Ordinarily, if a body of coal is allowed to heat spontaneously to a temperature approximating that of the boiling point of water, it will thereafter rise at an accelerated rate and "ignite," although this is not always true.

A few fires of spontaneous origin in English anthracites have been noted.[18] One of the writers is familiar with data on a fire in a refuse bank of Pennsylvania anthracite that undoubtedly originated spontaneously. However, fires of known spontaneous origin are rare in Pennsylvania anthracite. Calculations based on data of the type discussed above show that the conditions under which anthracite might ignite spontaneously are not met ordinarily in practical operations. The frequent reference in this paper to spontaneous heating is not because of any importance that could be attached to it in connection with fires in anthracite but because the principles involved serve so well to illustrate the behavior of fires, and knowledge of these principles is important in combative operations.

Effects of Oxidation on Composition of Coal

During the oxidation processes at temperatures up to about 350°C., the quantity of complex formed is proportional to that of oxygen consumed.[27,28] As the formation of this complex is attended by a net evolution of heat,[20] its presence decreases the thermal value of the coal. As previously stated, the complex can be removed by distillation and, in the ordinary methods of coal analysis, appears as volatile matter. The composition of this volatile matter, however, is quite different from that from fresh anthracite. The latter consists largely of hydrogen with smaller quantities of methane and oxides of carbon,[30] whereas the former is composed largely of oxides of carbon.[28] Since thermal value, volatile content, and ash content of anthracites can be correlated,[31] these facts may be applied directly to a determination of the extent to which a sample of coal may have been oxidized;[27] and, conversely, for simple cases, the effect of outdoor exposure on the composition and thermal value may be calculated beforehand.

If the temperature continues to rise and passes the "carbon monoxide inflection point,"[24] oxidation begins to remove the complex, and at about the ignition temper-

ature its removal is virtually complete.[28] This is not analogous to distillation, however, for in the latter complete removal of the complex from anthracite requires a temperature of around 1000°C.

Coal in an underground mine fire may reach such a high temperature as to clinker the ash. Chemical analyses or softening-temperature determinations of clinkers so found may be used to determine, if desired,

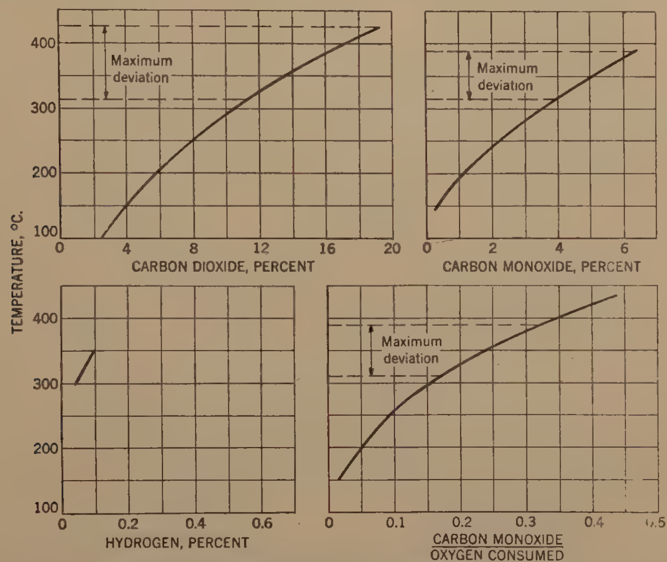

FIG. 3.—DATA FOR ESTIMATING TEMPERATURE OF ANTHRACITE WHEN OXIDIZED WITH AIR (AIR-AND-METHANE-FREE BASIS).

At temperatures above the carbon monoxide inflection point oxidation is so vigorous that the coal is thoroughly burned at the point where the oxygen first comes in contact with it, leaving a distinct exposure of ash; and this fact may be used to determine whether or not a coal has reached a temperature above the carbon monoxide inflection point, since at temperatures below this no ash appears except after a very considerable quantity of oxygen has been consumed.

Anthracites larger than about 20 mesh show the characteristic cracks caused by differential oxidation when oxidized at temperatures of 300° to 350°C.[22] These cracks are readily detectable with a hand lens, and may form a quick, qualitative means for determining whether a coal has been subjected to oxidation in this temperature range.

the minimum fusion temperature of the clinker, which will then indicate a temperature that the fire must have reached or passed at some stage of its existence.

Deductions based on the composition of coal from a fire area are somewhat of the nature of a post-mortem. However, as long as the fire lasts deductions must be largely from gas analysis.

RELATIONSHIP BETWEEN TEMPERATURE AND COMPOSITION OF A MINE-FIRE ATMOSPHERE

As stated previously, when coal first begins to oxidize the oxygen consumed is found in one solid and three gaseous products. Of the latter, two are fixed gases; *i.e.*, carbon dioxide and carbon monoxide. Up to a temperature of about 350°C. the ratio of carbon monoxide to oxygen consumed depends on the total quantity of

oxygen consumed. Since the total quantity of oxygen that can be consumed in a given time in this temperature range depends so greatly upon the temperature, these facts form the basis for an estimation not only found in the oxides of carbon.[25] This is shown in Fig. 4.

If hydrogen is found in the gases, the temperature probably is above 350°C. The ratio of carbon monoxide to oxygen con-

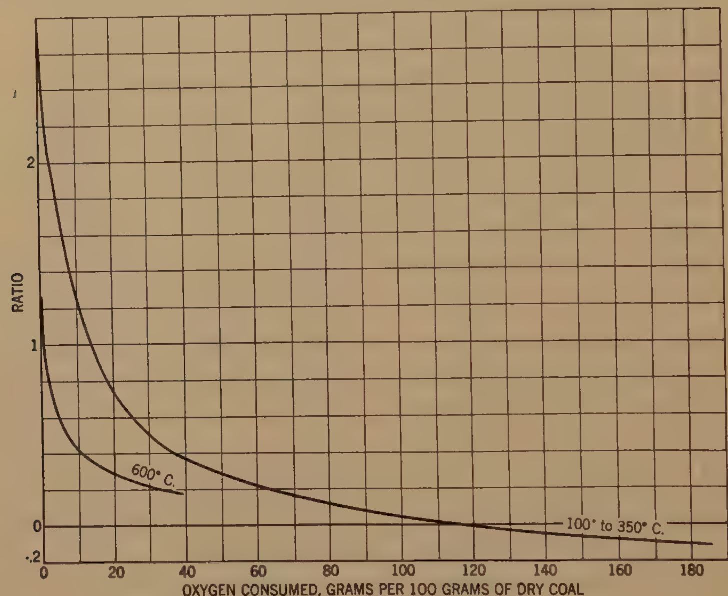

FIG. 4.—EFFECT OF EXTENT OF OXIDATION ON RATIO $\dfrac{\text{oxygen: (water + fixed oxygen)}}{\text{oxygen: (carbon dioxide + carbon monoxide)}}$

of the extent of oxidation but also of temperature in this temperature range (Fig. 3).[25] This ratio appears to be relatively independent of particle size and time of contact, whereas the oxygen concentration itself is useless for such a deduction because it is dependent upon both.

Hydrogen does not appear either in the oxidation or distillation products of anthracite until a temperature of 300° to 350°C. is reached; its absence is one necessary criterion, therefore, in establishing the fact that the temperature range will permit the ratios in Fig. 3 to apply.

Another method for estimating the extent of oxidation in this temperature range is from the ratio of the unaccounted-for fraction of the oxygen consumed (*i.e.*, to water and fixed oxygen) to the oxygen

sumed then becomes no longer applicable because the rate of oxidation of the carbon monoxide in contact with oxygen and coal is more rapid, and the ratio is affected by the surface area and time of contact. In this range (up to about 600°C.), the extent of oxidation above about 500°C. may be estimated roughly from the ratio of hydrogen to carbon in the oxidation products because, as previously stated, the compounds in coal with the higher hydrogen-carbon ratios are consumed first. This information may be useful in determining whether a fire is raging in a plentiful supply of virgin coal or the coal has been largely consumed, with a possibility that the fire may die for lack of fuel.

When a temperature of 545° to 660°C. is reached the carbon monoxide and hydrogen

formed burn directly in the gas phase if oxygen is present. Above this temperature, therefore, both the carbon monoxide and hydrogen may approach zero, depending upon the ratio of air to fuel. Some of these the quantity found in the gaseous products depending upon temperature, time of contact, and area of surface exposed. After the oxygen has been consumed the carbon dioxide in contact with coal is reduced by

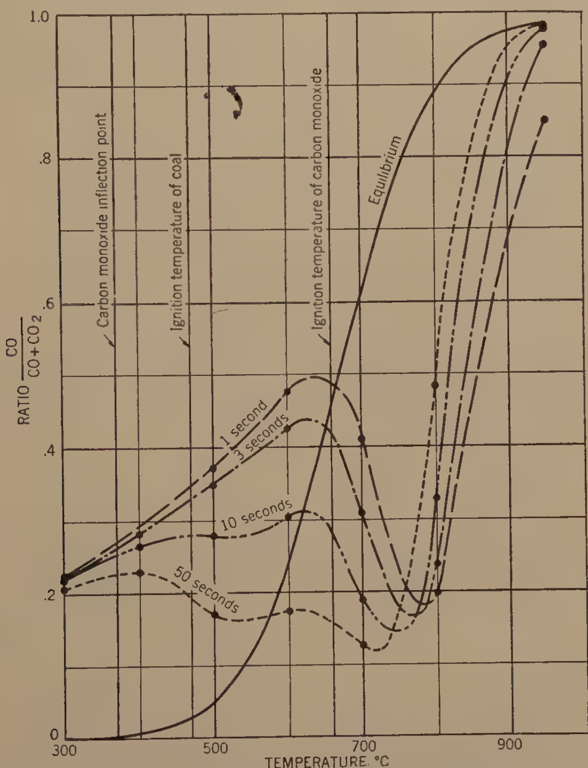

FIG. 5.—RATIOS CO TO CO + CO_2 FROM OXIDIZING ANTHRACITE; DRIED, PRIMARY AIR. Sample No. 16-B; size, 14 to 28 mesh.

relationships are shown in Fig. 5. An active fire with enough secondary air would show such a condition (no H, little CO, and much CO_2).

In the combustion of anthracite, when hydrogen and carbon monoxide are burned by secondary air the ratio of hydrogen to carbon monoxide has been found to be considerably below unity.[11,12] The other extreme is found in the pure distillation products, in which the ratio may be considerably above unity.[21]

At temperatures above 660°C. the carbon monoxide is burned in the gas phase as well as at the coal surfaces if oxygen is present, the producer-gas reaction, the rate of reduction increasing with temperature; or, to consider the hydrogen and water at the same time, the water-gas reaction should be used. Both equilibria must be satisfied, of course, if the reactions reach an equilibrium state. At temperatures below 660°C. the carbon monoxide in the primary oxidation products appears in greater quantity than the equilibrium state calls for. Thus, increase in time of contact or surface area causes a decline in the carbon monoxide below 660°C., while above this temperature there is first a decrease (due to direct burning of the monoxide by air) and then an

increase in carbon monoxide after the oxygen has been depleted. The result of this condition is a crossing of the equal-time curves, or an inversion effect, which appears to take place between 700° and 800°C. an actual sealed area in an anthracite fire. The two curves represent data from behind two different seals of the same fire. To be noted is the second hump in each curve, marked *A*.

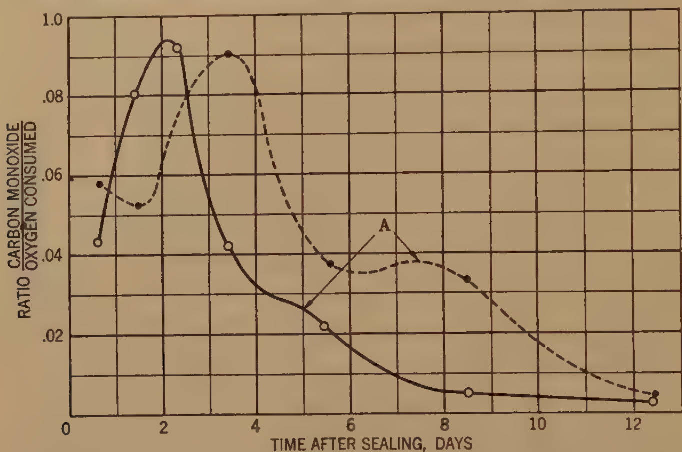

FIG. 6.—RATIOS CO TO OXYGEN CONSUMED FROM SEALED ANTHRACITE FIRE AREA.

These conditions are illustrated in Fig. 5, plotted from unpublished data of the authors. The same general picture is shown for the ratio of carbon monoxide to consumed oxygen.

If a fire area is well sealed the gases within the sealed area normally circulate within that area, producing the same effect as an increase in contact time. Because of oxygen depletion and heat dissipation, the effective temperature decreases. In these conditions the ratios $\frac{CO}{CO + CO_2}$ and $\frac{CO}{O_2 \text{ cons.}}$ would be expected to rise, reach a maximum somewhere above 700°C. and then decline. In the region between 500° and 660°C. a slight rise to a second but lower peak might be expected. This would be followed by a further decline to a nearly constant value. Complete disappearance of the carbon monoxide depends upon leakage or oxidation by bacteria; the latter, in turn, requiring oxygen for the reaction. Fig. 6 shows a plot of the ratio of carbon monoxide to oxygen consumed against time for

The greatest uncertainty in the use of gas analysis as a means of determining the behavior of a mine fire is probably caused by external factors. The percentage of carbon dioxide may be lowered in the gas phase by adsorption on the coal[29] or by solution in water, carbon dioxide being more readily adsorbed by anthracite and more readily dissolved in water than the other constituents. Both hydrogen and carbon monoxide may be oxidized by bacteria at ordinary temperatures if oxygen is present.[10] Then, there is always the question as to the representativeness of the samples. The composition may be; affected somewhat by other, apparently minor, factors. A high dilution of a fire gas with air, of course, reduces the accuracy of deductions therefrom.

The decomposition pressure of the ferrous sulphate formed by oxidation is dependent upon the temperature alone; therefore the concentration of sulphur dioxide in the gases may be used to estimate a minimum, but only a minimum, temperature. This limitation is due to its

high solubility in water, its probable high adsorption on the coal,[5] and the effect of dilution with air. The relationship is shown on Fig. 7.

Explosive Gas Mixtures

Methane is evolved from anthracite for some time after the fresh surfaces are exposed. It will accumulate in a sealed area and, unless oxygen is depleted rapidly enough, may contribute to the formation of explosive mixtures. It is imperative that no source of ignition shall be present when such a mixture is formed, which means that the fire itself should have cooled below the ignition temperature of the gas mixture.[14] Other possible sources of ignition create a dangerous condition until the composition of the gas mixture passes outside the inflammable range,[9] a condition usually brought about by continued increase in methane content and decrease in oxygen concentration. Fig. 8 is a plot of the inflammable conditions for the fire represented in Fig. 6. When a sealed area is reopened the inflammable range must be passed through again, but in reverse order.

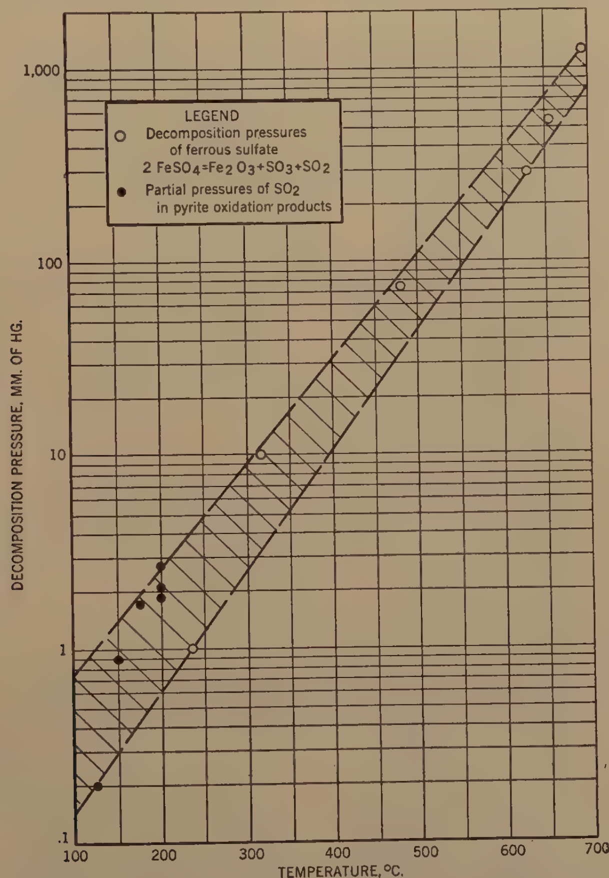

Fig. 7.—Decomposition pressures of ferrous sulphate.
From Chemical Engineers' Handbook, p. 355. John H. Perry, Editor in Chief. New York, 1934. McGraw-Hill Book Co.

From temperature estimates, gas analyses, and information concerning the physical environment of a fire, the chemist may

be able to predict whether or not the application of water is likely to cause the formation of enough hydrogen to create an explosive atmosphere.

Calculation of Leakage from Sealed Area

The quantity of air leaking into and out of a sealed area may be calculated approximately from the methane content, as follows:

From the estimated volume of the sealed area and the initial tangent to the methane-time curve, the rate of methane formation in the area can be computed. When the methane-time curve becomes horizontal, the rate of methane leaving the area is equal to the rate of methane entering the area. The accuracy of the computation depends, among other things, upon the constancy of the rate of methane evolution. If the leveling off takes place within a short time, it is possible that the methane evolution may remain relatively constant.[17] If the methane-time curve does not level off, and pressure increases, a tight seal is indicated.

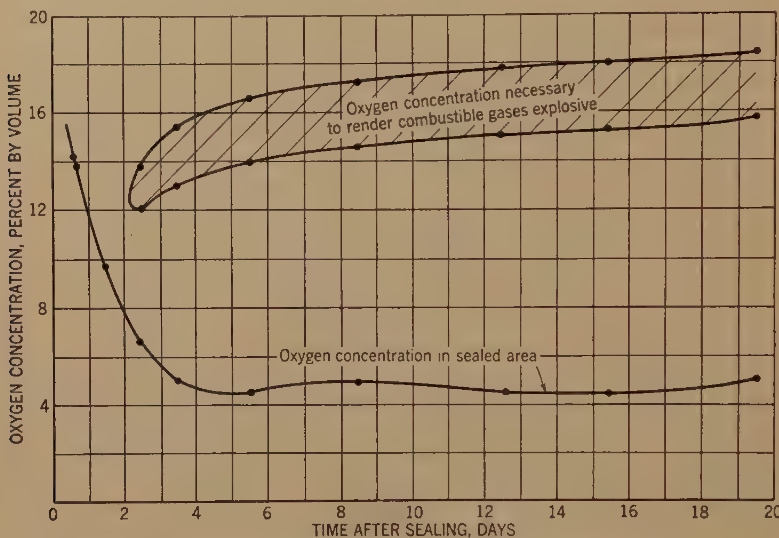

Fig. 8.—Plot of inflammable conditions for fire of Fig. 6.

When Is a Fire Dead?

If leakage into a sealed area is computed and the rate of methane evolution estimated, as previously discussed, the rate of oxygen change in the gases may serve as a means for estimating the amount of oxidation taking place behind the seals. For example, suppose that at the time a fire is thought to be extinguished the rate of methane evolution is estimated at 173,000 cu. ft. per day, and leakage out of the area is estimated at 352,000 cu. ft. per day. The oxygen content of the gases is, say, 5 per cent, and the tangent to the oxygen-time curve* is +0.065 per cent per day. In a sealed area of 6,500,000 cu. ft. the oxygen-consumption rate would be

$$0.209 \times (352{,}000 - 173{,}000) - 0.05 \times 352{,}000 - 6{,}500{,}000 \times 0.00065 = 15{,}600 \text{ cu. ft. per day.}$$

Coal and rock are very poor conductors of heat, and for a time after a sealed area has been opened close inspection is neces-

* The oxygen-time curve is a plot of the concentration of oxygen against time (or dates) after sealing.

sary to prevent the residual heat and increase in oxygen concentration attending the opening from causing a recurrence of fire (Fig. 2).

Quantitative knowledge of the decomposition pressures of limestone show that insofar as its ability to produce a blanket of CO_2 is concerned, its value at low temperatures is virtually nil, while at high temperatures its decomposition will have only a partial cooling effect. The main advantages to be derived from the use of powdered limestone depend upon its blanketing effect on the air movement and its physical ability to absorb some of the heat.

The direct application of chemical science to control of mine fires depends, as may be surmised, upon the knowledge and ingenuity of the chemist; with good equipment he should be able to contribute to the conservation of life and valuable natural resources.

References

1. G. Agde and G. Gölz: Researches on the Ignitibility of Brown Coals. *Braunkohle* (1933) **32**, 386.
2. G. Coles and J. I. Graham: Influence of Preheating Unoxidized and Partially Oxidized Coal upon Its Rate of Absorption of Oxygen. *Fuel* (1924) **3**, 384.
3. J. D. Davis and D. A. Reynolds: Spontaneous Heating of Coal. U. S. Bur. Mines *Tech. Paper* 409 (1928).
4. H. Fayol: Studies of the Alteration and Spontaneous Combustion of Coal Exposed to Air. *Bull.* de la Soc. de l'Industrie Minerale. (1879) [2] **8**, pt. 3.
5. J. I. Graham: Adsorption or Solution of Methane and Other Gases in Coal, Charcoal, and Other Materials. *Trans.* Inst. Min. Engrs. (1922) **62**, pt. 4, 298.
6. J. I. Graham: Permeability of Coal to Air or Gas—The Solubilities of Different Gases in Coal. *Trans.* Inst. Min. Engrs. (1917) **52**, pt. 3, 338.
7. J. I. Graham and W. E. Raybould: Spontaneous Firing of Coal with Special Reference to North Warwickshire Seams. *Trans.* Inst. Min. Engrs. (1932) **83**, 89.
8. A. Jäppelt: Self-inflammability of Brown-coal Coke and Its Cause and Its Cure. *Braunkohlen Arch.* (1928) **20**, 1.
9. G. W. Jones: Inflammability of Mixed Gases. U. S. Bur. Mines *Tech. Paper* 450 (1929).
10. G. W. Jones and G. S. Scott: Oxidation of Carbon Monoxide and Hydrogen by Bacteria. U. S. Bur. Mines *R.I.* 3466 (1939).
11. H. Kreisinger, C. E. Augustine and S. H. Katz: Low Rate Combustion in Fuel Beds of Hand Fired Furnaces. U. S. Bur. Mines *Tech. Paper* 139 (1918).
12. H. Kreisinger, F. K. Ovitz and C. E. Augustine: Combustion in the Fuel Bed of Hand Fired Furnaces. U. S. Bur. Mines *Tech. Paper* 137 (1917).
13. H. H. Lowry: The Chemical—Coal. *Ind. and Eng. Chem.* (1934) **26**, 133.
14. T. N. Mason and F. V. Tideswell: Gob Fires, I—Explosions in Sealed Off Areas in Non-Gassy Mines. Safety in Mines Research Board *Paper* 75 (1933).
15. G. E. McElroy: Some Observations on the Causes, Behavior, and Control of Fires in Steep-pitch Anthracite Mines. U. S. Bur. Mines *Inf. Circ.* 7025 (1938).
16. H. C. Porter and A. C. Fieldner: Weathering of the Pittsburgh Coal Bed at the Experimental Mine Near Bruceton, Pa. U. S. Bur. Mines *Tech. Paper* 35 (1914).
17. H. C. Porter and F. K. Ovitz: Escape of Gas from Coal. U. S. Bur. Mines *Tech. Paper* 2 (1911).
18. D. W. Rees: Is Spontaneous Combustion of Coal Attributable, in Part, to Oxidation of Occluded Gases? *Coal Age* (1926) **30**, 913.
19. F. E. Rhead and R. V. Wheeler: Mode of Combustion of Carbon. *Trans.* Chem. Soc. (1913) **103**, 461.
20. G. S. Scott: Heat Liberated in the Low-temperature Oxidation of Anthracite. U. S. Bur. Mines *Inf. Circ.* 7053 (1939).
21. G. S. Scott and G. W. Jones: Composition and Inflammability of Gaseous Distillation Products from Heated Anthracite. U. S. Bur. Mines *R.I.* 3378 (1938).
22. G. S. Scott and G. W. Jones: Effect of Particle Size on the Oxidation Rate of Anthracite. U. S. Bur. Mines *R.I.* 3546 (1941).
23. G. S. Scott and G. W. Jones: Oxidation of Anthracite—Effect of Time of Contact on the Concentration of Oxygen in the Effluent Gases. U. S. Bur. Mines *R.I.* 3405 (1938).
24. G. S. Scott and G. W. Jones: Oxidation of Anthracite—Liberation of Carbon Monoxide and Its Relation to Ignition Temperature. *Ind. and Eng. Chem.* (1937) **29**, 106.
25. G. S. Scott and G. W. Jones: Phenomena Observed during the Prolonged Oxidation of Anthracite. U. S. Bur. Mines *R.I.* 3504 (1940).
26. G. S. Scott and G. W. Jones: Significance of Low Temperature Oxidation Products in Anthracite Mine Fire Studies. *Ind. and Eng. Chem.* (1937) **29**, 822.
27. G. S. Scott, G. W. Jones and H. M. Cooper: Effect of Oxidation of Anthracite on Its Heating Value. *Ind. and Eng. Chem.* (1939) **31**, 1025.
28. G. S. Scott, G. W. Jones and H. M. Cooper: Effect of Oxidation on the Volatile Matter of Anthracite and Its Significance in Mine Fire Investigations. U. S. Bur. Mines *R.I.* 3398 (1938).
29. E. Sinkinson and H. G. Turner: Adsorption of Carbon Dioxide by Coal. *Ind. and Eng. Chem.* (1926) **18**, 602.
30. H. G. Turner and W. L. Keene: Volatile Matter of Pennsylvania Anthracite. *Ind. and Eng. Chem.* (1935) **27**, 1373.
31. H. G. Turner and G. S. Scott: Chart for Estimating B.t.u. of Anthracites from Proximate Analysis. Anthracite Inst. (1934).
32. R. V. Wheeler: The Oxidation and Ignition of Coal. *Jnl.* Chem. Soc. (1918) **113**, 945.
33. T. F. Winmill: The Absorption of Oxygen by Coal, VI—The Rate of Spontaneous Heating of Coal. *Trans.* Inst. Min. Engrs. (1914) **48**, 535.

Thermodynamics and Coal Formation

By Walter Fuchs,* Member A.I.M.E.

(New York Meeting, February 1941)

It is now generally conceded that coal is the product of deposition and transformation of debris of forests and swamps.[29] Ample data are available to illustrate the metamorphosis of biochemical substances through biological action; other data show the progressive regional metamorphism existing in coal beds;[25] and numerous examples of change effected by intrusion of igneous dikes into coal deposits have been established.[23] However, often the relative importance of the biological and the geological factor in determining the character of the various natural coals is a debatable question.

Since both biological and geological factors are ever present in nature, the problem, at first glance, would appear to resemble certain probability problems: specifically the simultaneous toss of two coins, with the biological factor as heads and the geological as tails. The probability is 2:1 for the combined effect of both factors, as compared to the exclusive effect of either one, if the *a priori* probability is the same for each factor. This condition can be analyzed by a thermodynamic treatment of data, which may be obtained from a consideration of pertinent biological, chemical, geophysical and geological facts.

Synopsis of Biological and Chemical Facts

The biosphere is not sterile; i.e., microorganisms are always present. Microorganisms have been established in the ocean deep, under the ice of the polar regions, in the sand of the deserts, and in swamps far below the surface. The various biological media may be acid, neutral, or alkaline, and they may or may not be readily accessible to air. Microorganisms are encountered almost everywhere, except where the temperature is not within the range tolerable by microbial life. Of course, with changing conditions, the type of microorganisms will change.

All chemical products of life—proteins, fats, carbohydrates, lignins, waxes, resins—may serve as a habitat of microorganisms. In the decomposition of a medium, some microorganisms macerate organic matter mechanically and remove certain constituents for their own nutrition; others decompose a large part of the organic matter but may leave certain constituents undecomposed. Some attack only certain specific ingredients of the organic matter. All of them build up cell substance. The nitrogen content of the medium largely determines the degree of decomposition. Below 1.7 per cent nitrogen, growth of microorganisms and hence decomposition of organic material is retarded; consequently organic material will accumulate.[17,32] Such accumulations of organic material may give coal eventually.

Coals differ in type and rank. The nature of the original vegetation and the conditions attending the transformation of the accumulated debris determine the type of coal such as bright, splint, cannel, and algal or boghead coals. Chemical analysis of representative samples of a given type of

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[29] References are at the end of the paper.

coal reveals large differences in ultimate and proximate composition. "The higher rank coals are classified according to fixed carbon on the dry basis and the lower rank coals according to B.t.u. on the moist basis."[4]

A few of the chemical components of plants are found in solid fuels: e.g., cellulose is common in peats and not unusual in brown coals;[8] and sporonin and other plant substances are reported to occur in bituminous coals.[7] However, the bulk of most coals consists of substances not present in living plants; and this phenomenon may be explained in two different ways. If it is conceded that the original plant debris has served as a habitat of microorganisms, an explanation of the change of vegetal material to coal may be based on the ability of microorganisms to effect changes in the media. If the biological factor is considered to be insignificant, heat and pressure should have been the agents involved in this transformation.

Synopsis of Geophysical and Geological Facts

The development of minor heat quantities in the earth's crust has been ascribed, apart from volcanism, to chemical reactions, friction on fault planes, porous plug expansion, heat developed by compression and shear, and many other causes. Van Orstrand[21] has presented a rough estimate of the average temperature gradient in the sedimentary areas of the United States: "The reciprocal of the average gradient is almost certainly greater than 60 ft. per degree Fahrenheit (32.9 m. per degree centigrade) and possibly as large as 110 ft. per degree Fahrenheit (60.4 m. per degree centigrade)." With a value of 50 m. per degree centigrade, a covering layer of 1000 m., or roughly 3000 ft., in this area, would account for a temperature rise of 20°C.

A deposit within the earth's crust may be compressed either by weight through depth of burial, or by horizontal forces accompanying orogeny. From a knowledge of the specific gravity and the height of the overlying strata, the pressure due to weight is easily computed. Jeffreys[16] points out that the pressure due to the weight of the Himalayas amounts to about 10^9 dynes (1000 atmospheres) per sq. cm. The weight of mountain ranges comparable to the Rockies or the Alps would cause much smaller pressures. The magnitude of horizontal forces in the earth's crust has also been estimated by various investigators, who found themselves in substantial agreement. According to Jeffreys,[16] the crust of the earth can be taken to transmit stresses perfectly for any distance, and failure takes place where the strength of the rocks is exceeded. That strength is of the order of 10^9 dynes (1000 atm.) per sq. cm. Gutenberg[12] estimates the maximum value of horizontal forces in the earth's crust caused by deviation from hydrostatic equilibrium to be 10^9 dynes (1000 atm.) per sq. cm., while those due to vertical movement of blocks in the earth's crust are judged to amount to only 10^4 dynes (0.01 atm.) per sq. centimeter.

Heat in significant amounts may penetrate into a coal deposit either through intrusion of igneous rocks or on account of location at great geothermic depth. Examples of intense change of a coal by contact with intrusive igneous rocks are numerous. These changes extend usually over only a short distance from the point of contact; and samples taken along that distance show a definite beneficiation; i.e., in terms of coal classification, an increase of rank. Results of investigation of 16 different cases are compiled in Fuchs[7] (Table 151, p. 385). As the point of contact is approached the products resemble natural cokes rather than coals of higher rank; and it may not be easy to decide without a thorough investigation whether a thermal transformation product with, for instance, 18 per cent volatile matter should be classified

as a low-volatile bituminous coal or as a low-temperature coke.

A rough empirical rule concerning the relationship between rank and geothermic depth—known as Hilt's rule—refers to the observation that in many seams the volatile-matter content decreases with depth.[7] This rule is not without exceptions, and where it holds no simple relationship between depth and rank is discernible. This is the conclusion reached by Petrascheck[23] on the basis of a careful examination of all available data. Another conclusion reached by this author is that in the formation of bituminous coals apart from cases of intruded igneous rocks the high temperatures assumed in the older literature have never been reached.

Within a given coal bed, a change of rank in a lateral direction is a common phenomenon; e.g., Campbell[3] studied the Pittsburgh coal bed in the northern part of the Appalachian coal field. He compared coal samples taken from large mines on an almost continuous line of outcrop, 130 miles long and extending at right angles to the major axis of the trough containing the coal-bearing rocks; the fixed carbon increased gradually from 43.8 per cent to 73.2 per cent. Stadnichenko[25] found a progressive change of chemical and physical characteristics in the Lower Kittanning seam in an eastward direction, the coal increasing from the Allegheny river to the Allegheny front from 58.6 per cent fixed carbon to 84.7 per cent fixed carbon. These and other examples have been explained by assuming climatic differences, differences of original vegetation, changing geothermic depth, depth of burial, and thrust pressure. Objections have been raised against every one of these explanations.

Geological age.—i.e., time—apparently is without influence upon rank. Typical brown coals have been found in formations of Lower Carboniferous age (Moscow and Ukraine coal basins); on the other hand, bituminous, semibituminous and anthracitic coals are not uncommon in Mesozoic and Tertiary formations (Canada, United States, Siberia). The idea that differences in age offer an explanation of the various ranks of coal has been largely abandoned. The brown coals found in very old carboniferous formations furnish direct evidence to show that the mere lapse of time caused no change of rank. The assumption that time was a rank conditioning factor in other coal deposits (e.g., by virtue of slow reaction rates) is not supported by direct evidence.

Discussion of Geological Field Evidence

Campbell[3] investigated a cross section of the Pittsburgh coal bed beginning at a point in Ohio about 25 miles west of Wheeling, W. Va., and continuing southeastward to the vicinity of Cumberland, Md. He presents a diagram in which "a line drawn at the top of the fixed carbon content ascends with remarkable regularity from one end of the section to the other, showing that the chemical composition of the coal changes gradually and steadily along this line for a distance of 130 miles." This is indeed correct: a least-square treatment of Campbell's data showed that the relationship between distance x from an arbitrary origin and fixed carbon content y is well represented by the equation:

$$y = 42.6 + 0.23x(\pm 1\%)$$

Experimental and computed data are presented in Table 1.

There are no residuals to be explained, the derived equation being self-sufficient. It matters little that Campbell traces the observed change of rank to base deformations of the coal measures from mile 75 to mile 130. The fact that the slope of the line referred to by Campbell shows no discontinuity proves conclusively that base deformation was no factor in changing the rank of coal. The slope of the curve would be the same if deformation had occurred

TABLE 1.—*Fixed Carbon Contents in the Pittsburgh Coal Bed*

No.	Location	Distance from No. 1, Miles	Fixed Carbon, Per Cent	
			Experimental	Computed
1	Ohio	0	43.8	42.6
2	Wheeling, W. Va.	23.3	48.3	47.9
3	West Brownsville, Pa.	63.3	55.9	57.1
4	Fairchance, Pa.	74.4	58.2	59.7
5	Meyersdale, Pa.	114.4	67.8	68.9
6	Frostburg, Md.	130.0	73.2	72.5

from mile 1 to mile 75, or from mile 1 to mile 130, or not at all. Additional factual refutation of the theory supported by Campbell is contained in the work of Heck[13] concerning the Sewell coal seam in southern West Virginia, and other seams below it. The carbon ratio of the Sewell coal shows a progressive change from 71 in Greenbrier County to 81 in southwestern Fayette County, a distance of 40 miles. The coal in Greenbrier County is stated to have suffered greater structural distortion than that in Fayette County; relief of stress by overthrusting is considered and rejected. According to Heck, the strata deposited over the Sewell bed in southwestern Fayette County were at least 1000 ft. thicker than those deposited on the same bed in Greenbrier County. Therefore, he concludes that depth of burial was the main reason of the metamorphism, at least up to the rank of semianthracite.

The assumption that depth of burial is a rank-determining factor has been considered and rejected by Stadnichenko.[25] In a study of the Lower Kittanning coal bed of western Pennsylvania, coals in the deepest part of the bituminous basin, beneath the maximum thickness of the Permian, in southwest Pennsylvania were found to be less metamorphosed:

> If the advancing carbonization is due to increase in thickness of cover in the same direction, . . . there should be a thickening of post-Kittanning rocks toward the Allegheny Front. On the other hand, the cited geological reports fail to show any important thickening of the Conemaugh and Monongahela formations toward the eastern border of the bituminous field, or in the Broad Top field to the eastward. In short, the aggregate of the superposed strata is not now, and apparently never has been, appreciably greater at the Allegheny Front along the Windber district than in the central area of the syncline.

The facts established by geological field work are indubitable, but the offered explanations are contested. Thermodynamic analysis of the underlying chemical problems should do much to clarify the situation.

SYSTEMATICS AND THERMODYNAMICS OF CHEMICAL REACTIONS

The chemical reactions in organic systems, diversified as they are, may be classified under two general headings, decomposition and synthesis. With a given type of molecule, or atomic arrangement, reactions such as hydrolysis or cracking would tend to decompose the given structure, polymerization or condensation would build up larger structures, while oxidation and reduction, inextricably linked together, may point the way in either direction. Essentially the same simplification of the topic is accomplished if all the reactions that are actually or potentially involved in coal formation are considered as making or breaking one or more of the following linkages: C—O, C—N, C—H, C—C. The final result of a chemical reaction is an equilibrium state that can be predicted, at least in principle, by the application of thermodynamic reasoning. The speed of attaining that final state of equilibrium cannot be predicted at the present time. If a thermodynamic possibility does not coincide with reality, studies of reaction rates and catalytic influences are necessary for a full understanding of the situation.

In the following sections, the thermodynamic analysis of the chemical systems that participate in coal formation is accom-

plished by the use of the free-energy function. It has been said that:

> The study of free energy affords the only true measure of chemical affinity, and although, when the free energies of all the substances involved in a given reaction are known, it may still be impossible to predict the rate of the reaction, it will be possible to state in advance in what direction and to what extent the process can ultimately occur.[19]

Free energies may be obtained from the fundamental relations 1 and 2, where ΔH

$$F = H - TS \qquad [1]$$
$$\Delta F = \Delta H - T\Delta S \qquad [2]$$

denotes the heat absorbed in a chemical reaction at constant pressure and ΔF the change in free energy in a chemical reaction at constant pressure and constant temperature. The computation of ΔF from Eq. 2 involves thermal measurements down to very low temperatures and the application of the third law of thermodynamics. However, for the purpose of this investigation, sufficiently accurate values may be obtained by the use of Eq. 3,

$$\Delta F^0 = -RT \ln K = -n\mathfrak{F}E^0 \qquad [3]$$

which summarizes the relationship between equilibrium constant K, electromotive force E of a reversible reaction, and ΔF^0; i.e. ΔF referred to elements and compounds in their commonest ("standard") states at the temperature T, solutes in molal solutions, and gases at 1 atm.* A convenient and more general relationship is given in Eq. 4:

$$\Delta F = -RT \ln (K/Q) \qquad [4]$$

"where Q has the same form as the equilibrium constant, but the concentrations refer to the values of the substances in their initial and final states. If these are unity, then Q is unity, and the free energy is ΔF^0."[18] "If ΔF^0_{298} has a large positive value, then we know that the reaction would not occur at this temperature to any measurable extent; if ΔF^0_{298} has a large negative value the reaction may run 'completely,' that is, may run so nearly to an end that no analytical methods would indicate the existence of an equilibrium. If the numerical value of ΔF^0_{298} is small, either positive or negative, the reaction may usually be led to proceed in either direction, by a suitable choice of concentrations and partial pressures."[19] For more detail, the reader is referred to the literature.[19,22]

Specific Reacting Systems

In presence of abundant water, hydrolytic processes are the most likely ones to occur, and the easiest ones to accomplish. Hydrolytic processes are apt to break down fats, proteins, and polysaccharides by the severance, in each case, of a characteristic linkage: the ester linkage in fats, the peptide linkage in proteins, the saccharide linkage in polysaccharides. Eq. 5 is descriptive of the hydrolysis of the ester linkage;

$$\text{R.COO.R}' + H_2O = \text{R.COOH} + \text{R}'\text{.OH}; \quad \Delta F^0_{298} = +900 \text{ cal.}; \quad \Delta F^0_{373} = +1000 \text{ cal.} \qquad [5]$$

R and R′ denote normal saturated aliphatic chains. This generalized treatment is predicated upon the fact that in a considerable number of specific cases the equilibrium constants of Eq. 5 are all about the same in magnitude and show little change with temperature.[22]

In a similar manner, general equations to represent free energies of making or breaking other types of atomic linkages may be derived.[2] Estimates of the free-energy changes in the hydrolysis of peptide linkages (Eq. 6), and saccharide linkages (Eq. 7), have been made; they were based upon data referring to heats of reactions and free-energy changes in the hydrolysis of urea and asparagin, of acetals and ethers. All

* In view of this definition of ΔF^0, the customary subscripts *s*, *l*, *aqu*, and *g* have been omitted in the following discussion; the reader should have no difficulty in recognizing the standard states.

these data indicate that the free-energy changes in reactions 6 and 7 are of the same order of magnitude as those in reaction 5.

$$NH_2R.CO.NH.R.COOH + H_2O = 2NH_2R.COOH \quad [6]$$

$$C_5H_{10}O_5CH.O.CHC_5H_{10}O_5 + H_2O = 2C_6H_{12}O_6 \quad [7]$$

It is interesting to note that the three biologically important linkages are thermodynamically equivalent and can be closed and opened with equal ease.

In dealing with specific cases, we must consider the number of hydrolyzable linkages per molecule and the ratio organic compound: water. Eq. 8 describes the hydrolysis of a fat with three ester linkages; in this equation the ratio fat: water is 100:6. At a ratio of 1:1, ΔF^0_{298} is negative,

$$(C_{17}H_{35}COO)_3C_3H_5 + 3H_2O = 3C_{17}H_{35}COOH + C_3H_8O_3 \quad [8]$$

the driving force of the reaction being further increased because both ester and acid are insoluble in water while the solution of glycerol in water causes a decrease of free energy. In other words, in a 50 per cent aqueous fat emulsion saponification is thermodynamically favored.

A protein may contain, for instance, from 100 to 1000 peptide linkages, and the number of saccharide linkages in compounds such as starch and cellulose is of the same order of magnitude. Yet, in presence of excess water, starch and proteins are rather easily hydrolyzed while cellulose does not hydrolyze readily. This phenomenon perhaps is due to the fact that proteins and starches in contrast to cellulose disperse rather easily in aqueous media. In a heterogeneous system, reaction rates are conceivably much lower than in a homogeneous system, although over-all thermodynamic conditions may be essentially the same. As an illustration of this, Eq. 9, which is descriptive of a complete cellulose hydrolysis, is tentatively analyzed.

$$(C_6H_{10}O_5)_n + (n - 1)H_2O = nC_6H_{12}O_6 \quad [9]$$

For a ratio of cellulose: water 9:1, the breaking of n glucosidic linkages ($100 > n > 1000$) points to a ΔF^0_{298} from +100.000 to +1.000.000 cal. In other words, slightly moistened cellulose at 25°C. is an entirely stable substance. At 25°C., with a ratio of 1:1 for cellulose: water, ΔF^0_{298} is probably still positive, though greatly reduced by the thermal effect of the solution of hydrolytic products. At a slightly elevated temperature and at a ratio of 1:10 for cellulose: water—that is, in presence of abundant water—cellulose hydrolysis is thermodynamicly favored, especially if the concentration of the products of hydrolysis is kept at a minimum; e.g., by a thriving population of microorganisms.

Typical oxidation-reduction processes are exemplified in Eqs. 10, 11 and 12, which refer to the oxidation and reduction of acetaldehyde. It is easily seen that all these reactions are thermodynamically probable. Whether oxidation or reduction will occur depends, of course, on the medium, oxidation being favored in an oxidizing medium and reduction in a reducing one.

$$2CH_3CHO + O_2 = 2CH_3COOH; \quad \Delta F^0_{298} = -12500 \text{ cal.} \quad [10]$$

$$CH_3CHO + H_2 = CH_3CH_2OH; \quad \Delta F^0_{298} = -8320 \text{ cal.} \quad [11]$$

$$CH_3CH_2OH + H_2 = CH_3CH_3 + H_2O; \quad \Delta F^0_{298} = -27060 \text{ cal.} \quad [12]$$

$$n\text{—}C_6H_{14} = n\text{—}C_6H_{12} + H_2; \quad \Delta F^0_{298} = +20000 \text{ cal.} \quad [13]$$

$$n\text{—}C_7H_{16} = n\text{—}C_6H_{12} + CH_4; \quad \Delta F^0_{298} = +5700 \text{ cal.} \quad [14]$$

$$C_6H_5COOH = C_6H_6 + CO_2; \quad \Delta F^0_{298} = -4760 \text{ cal.} \quad [15]$$

Eqs. 13, 14 and 15 illustrate the breaking of C—H and C—C linkages. Reactions such as 13 and 14 play an important part in the transformation of saturated aliphatic compounds into unsaturated cyclic compounds. This conversion may proceed either through dehydrogenation and subsequent cyclization of intermediate olefinic substances, or through ring closure and subsequent

dehydrogenation. Available data refer mostly to hydrocarbons.

Parks and Huffman[22] state:

> . . . that the paraffins and polymethylenes are relatively the more stable hydrocarbons at low temperatures (i.e., below 500°K.), while the aromatics and olefins are more stable at high temperatures. However, in no case save that of acetylene does the stability increase with temperature; and above 850°K. no hydrocarbon is thermodynamically stable with respect to the elements.

Francis[6] has discussed the conversion of paraffins into olefins as follows:

> . . . in order to make ΔF^0 negative, the temperature or the number of carbon atoms, or both, must be fairly high—conditions which favor the more profound disruption of the molecule. Likewise, the partial removal of hydrogen by means of atmospheric or other form of oxygen with the purpose of obtaining olefins . . . is possible thermodynamically because of the high free energy of formation of steam; but actually the chance of success would be very small because the products would be much more reactive with the oxidizing agent than was the starting material.

Reactions 16 and 17 refer to ring closure and subsequent dehydrogenation of hexane to benzene. Obviously, both reactions will be impeded by increased pressure, but favored by rising temperature. In Eq. 17, ΔF^0 becomes negative above 544°K.

$$C_6H_{14} = C_6H_{12} + H_2; \quad \Delta F^0_{298} = +7000 \text{ cal.} \qquad [16]$$

$$C_6H_{12} = C_6H_6 + 3H_2; \quad \Delta F^0 = +51500 - 94.7T \text{ cal.} \qquad [17]$$

Rather striking results are obtained by an analysis of the thermodynamic possibilities of the transformation of carbohydrates into cyclic substances. Eq. 18 refers to the hypothetical transformation of sucrose into naphthalene, while Eq. 19

$$C_{12}H_{22}O_{11} = C_{10}H_8 + 2CO_2 + 7H_2O; \quad \Delta F^0_{298} = -163{,}620 \text{ cal.} \qquad [18]$$

$$C_{12}H_{22}O_{11} = C_{10}H_6.OH.COOH + CO_2 + 6H_2O; \quad \Delta F^0_{298} = -295{,}000 \text{ cal.} \qquad [19]$$

deals with the hypothetical production of naphthalene hydroxycarboxylic acid from sucrose. ΔH for these reactions amounts to roughly −200,000 and −350,000 cal., respectively. In such strongly exothermic reactions the temperature will rise to a value at which naphthalene will decompose into the elements carbon and hydrogen, or give products resembling asphalts and cokes. In the transformation of complex carbohydrates such as cellulose into cyclic compounds, both ΔF and ΔH will probably be of the same order of magnitude as in Eq. 18 if a similar ratio of complexity between reagents and reactants is maintained.

If Eqs. 18 and 19 are considered as resulting from a sequence of steps, the first step probably will be the removal of one molecule of water from one molecule of carbohydrate. This reaction will occasion the release of roughly 60,000 cal. into carbonaceous material with a low specific heat; e.g., 0.3 cal. per gram. Hence, once reaction has started, rise of temperature and acceleration of reaction appear to be inevitable. Of course, in the laboratory, the heat created by an exothermic reaction may be dissipated, e.g., by carrying out a small-scale decomposition at elevated temperature in a large body of water or metal.

These conclusions are in line with a variety of observations; for instance, in cases of self-ignition of haystacks, coke- or charcoal-like "haycoals" are known to result from exothermic reactions of cellulosic material. From several laboratories, so-called "aromatizations" of cellulose have been reported; i.e., exothermic decompositions under conditions favorable to dissipation of the heat of reaction. Processes related to self-ignition have been assumed by some authors (and rejected by others) in explaining the origin of fusain; otherwise, they seem to have no bearing on coal formation.

Influence of Heat and Pressure

In this section, an attempt is made to estimate the energy that was available in

form of heat and compression work in a deposit of solid fuel heated temporarily to an elevated temperature. Influx of heat can be estimated on the basis of a knowledge of heat capacity and maximum temperature of the heated body.

The heat capacity of coal on a dry basis is around 0.3 cal. per gram, and on the basis of a moisture content of 50 per cent, around 0.65 cal. per gram; intermediate values may be assigned to deposits with a moisture content between zero and 50 per cent. The maximum temperature to which a coal deposit has been exposed may be estimated by establishing reference or indicator substances with known thermic properties. In brown-coal deposits, the presence of amber[9] and of cellulose[8] seems to preclude temperatures above 200°C. Only slightly higher maximum temperatures have been deduced for a number of deposits of bituminous coals, partly on the basis of geothermal depth, partly on the basis of indicator substances such as chlorophyll and hemin derivatives,[30] sporonin, and others.[7] It should be emphasized that though the above-mentioned temperatures cannot possibly have been exceeded, they need not necessarily have been reached. G. Thiessen[28] has offered "sufficient evidence to prove that the maximum temperatures during the formation of bituminous coals were not greatly in excess of those found today in coal beds, certainly not much greater than 100°C." However, for the sake of the following argument, a maximum temperature of 250°C. is assumed.

Let a coal deposit with a temperature of 25°C. and a moisture content of 50 per cent be placed in a huge heat reservoir with a temperature of 250°C. A large portion of the entering heat will be consumed in bringing the water up to 100°C., in promoting hydrolytic processes, and finally in evaporating the remainder of the water; 45 cal. per gram of dry fuel will be consumed in bringing the fuel deposit from 100°C. to 250°C. From this point on, only endothermic reactions could draw additional amounts of heat isothermally into the deposit; they are unlikely below 500°C., a temperature that is easily attained only through intrusion of igneous rocks.

The melting points of the minerals that make up volcanic dikes in coal deposits point to sustained temperatures of the order of magnitude of 1000°C. Endothermic reactions will be induced in that range of temperature with carbon or materials approaching carbon as ultimate reactants. This effect is known to fade out in a short distance, and there seems to be no relation in time or in space between such localized phenomena and the gradual change of rank observed in many coal beds over appreciable distances.

The maximum potential pressure upon a coal deposit has been estimated to be about 1000 atm. per sq. cm. Compression will first result in mechanical compaction of the solid fuel, the limit of compaction being set by the crushing strength of the fuel. The crushing strength of bituminous coal is approximately 70 atm. per sq. cm., and of anthracite, 280 atm. per sq. cm. Beyond these limits, coals may suffer plastic deformations, or more probably, they will crumble and become friable. These considerations should be helpful in distinguishing actual and potential compression. However, let it be assumed, for the sake of the argument, that during coal formation pressures of the order of 10,000 atm. per sq. cm. have been realized. In a homogeneous system, equilibrium shifts with rising pressure toward the side with the smaller number of molecules. In a condensed heterogeneous system, rising pressure will cause phenomena making for decreasing volume; e.g., melting points will be lowered, boiling points will be raised, and simple polymerizations will be favored. Hence, only small changes of proximate analysis, but no significant change of ultimate composition or rank of coal may result from compression.

These conclusions, though hardly in line with frequently encountered opinions, are borne out by the results of recent experimental investigations. Hoffman[15] exposed various coal samples for 4 days to a pressure of 18,000 kg. per sq. cm. and noted an average decrease in volatile matter by 1.5 per cent. The volatile-matter content of a gas coal that had been heated to 180°C. during compression decreased by 1.7 per cent. On the other hand, Trifonow and Toschew[31] compressed nine different coal samples by applying a pressure of 10,000 kg. per sq. cm. They observed a decrease of the so-called solid bitumen by a few tenths of one per cent and a slight increase of the total volatile matter.

Compression may, conceivably, cause a rise of temperature in the compressed system. Eq. 20 may be used for a quantitative estimate; it is transformed by the use of the thermodynamic relation 21 into Eq. 22 and solved as shown:[1]

$$dT = \left(\frac{\partial T}{\partial P}\right)_S dP \qquad [20]$$

$$\left(\frac{\partial T}{\partial P}\right)_S = \left(\frac{\partial V}{\partial T}\right)_P \frac{T}{C_P} = \frac{\alpha T}{C_P} \qquad [21]$$

$$\Delta T = \frac{\alpha T}{C_P}\int_O^P dP \qquad [22]$$

Substituting a starting temperature of 25°C., a maximum compression of 10,000 kg. per sq. cm., a specific heat of 0.3 cal. per gram of coal, a volume expansivity α of 0.2×10^{-4} and a specific gravity of 1.3, we obtain Eq. 23.

$$\Delta T = \frac{\alpha T}{C_P} \times 10^4 = \frac{298}{0.3 \times 42.66} \times \frac{20 \times 10^{-6}}{1.3} \times 10^4 = 3.5°\text{C.} \qquad [23]$$

This value is obviously negligible for the problem under consideration.*

Even if it is admitted that neither the normally available amount of heat, nor an excessive amount of compression work, can possibly account for the change in rank from a lignitic to an anthracitic coal, it is still conceivable that the phenomenon is due to the combined effect of both factors. It should be understood that the so-called volatile-matter content, which is the most important criterion of rank, is actually indicative of the thermal stability of the various atomic arrangements present in coal. The fact that upon heating one coal gives more volatile split products than another is in all probability sufficiently explained by a larger oxygen content: structures comprising oxygen break up more easily than hydrocarbon structures. Assuming that the change of rank was accomplished in nature by heat and pressure, we conclude that pressure was either immaterial or counteracting the effect of heat, at least at temperatures below 300°C.

As regards still higher temperatures, conclusions in line with those mentioned, p. 225, may be derived from results of experimental studies concerning the effect of pressure in coal carbonization. Fischer, Bahr and Sustmann[5] carbonized various lignitic and bituminous coals at 600°C. under gas pressures up to 100 atm. and established that with increasing pressure distillation of the bituminous tar-forming components of coal was increasingly obstructed. This obstruction resulted in certain cases in increased decomposition of the tar-forming substances, decreased tar yield, increased gas yield, and increased yield of coke with improved mechanical properties. These phenomena are closely connected with the percentage of bituminous substances, and virtually independent of rank; e.g., carbonization of a lignite with 65 per cent volatile matter, and of a coal with 10 per cent volatile matter, both coals having a low content of bituminous substances, gave identical results whether pressure was applied or not. On the other hand, different observations were made with coals that normally gave high

* By shearing stress, considerable but strictly localized temperature elevations may be created.

tar yields—a bituminous coal with 40 per cent volatile matter, described as giving normally 68.8 per cent coke, 13 per cent tar, and 7.3 liters of gas, gave 71.5 per cent coke, 2.2 per cent tar, and 13 liters of gas when carbonized under a pressure of 100 atm.; and a lignite giving normally 42.3 per cent coke, 22 per cent tar, and 11.5 liters of gas, gave 46.9 per cent coke, 8 per cent tar, and 21.0 liters of gas when carbonized under 50 atmospheres.

Oxidation-reduction Potentials

If the progressive reduction of a peatlike solid into coals of various ranks was not accomplished by heat and pressure, other factors must have taken part in the change. In this section, the oxidation-reduction potential (redox potential) is discussed because it offers a possibility of either decreasing or increasing the oxygen content of a suitable carbonaceous material at low temperature and pressure.

It has been noted that "bacterial cultures are nearly always strongly reducing systems."[14] As explained by Hewitt and other students of the subject, this indicates that microorganisms are able to utilize free oxygen, combined oxygen, hydrogen acceptors (or electron acceptors), or any suitable type of oxidizing agent. The underlying reactions, of course, are favored by the presence of suitable enzymatic systems. The actual value of the redox potential is determined by three factors: the presence of substances with a tendency to take up or give off electrons; the ratio of oxidant to reductant; and the pH of the medium. The electromotive force E is given by Eq. 24, where n indicates the number of charges passing from the reductant to the oxidant and C refers to the product of the active concentration on the two sides of the oxidation-reduction equation.

$$E = E^0 + \frac{RT}{n\mathfrak{F}} \times \ln \frac{C_{\text{red.}}}{C_{\text{ox.}}} \qquad [24]$$

For example, in the reduction of quinone, the oxidation-reduction equation is Eq. 25, the electromotive force being given by Eq. 26

$$C_6H_4O_2 + 2H^+ + 2e^- = C_6H_4(OH)_2 \qquad [25]$$

$$E = E^0 + \frac{RT}{2\mathfrak{F}} \times \ln \frac{[C_6H_4(OH)_2]}{[C_6H_4O_2][H^+]^2} \qquad [26]$$

It is easily seen that E^0 indicates the electrode potential when 50 per cent of the oxidized form has been reduced at unit concentration of hydrogen ion.

An example of an oxidation-reduction reaction known to occur in nature on a large scale under the influence of microorganisms is the mutual transformation of ammonia, nitrous acid, and nitric acid in soils. Eqs. 27, 28 and 29 are examples to the point.

$$HNO_3 + 2H^+ + 2e^- = HNO_2 + H_2O; \quad \Delta F^0_{298} = -43130 \text{ cal.}; \quad E^0 = +0.93 \text{ volt} \qquad [27]$$

$$HNO_2 + 4H^+ + 4e^- = NH_2OH + H_2O; \quad \Delta F^0_{298} = -58440 \text{ cal.}; \quad E^0 = +0.63 \text{ volt} \qquad [28]$$

$$2NH_2OH + 2OH^- = 2NH_3 + 1.5O_2 + H_2O + 2e^-; \quad \Delta F^0_{298} = +35650 \text{ cal.}; \quad E^0 = -0.77 \text{ volt.} \qquad [29]$$

In presence of organic compounds capable of ionization, the redox potential of the medium offers the possibility of reducing, under anaerobic conditions, the oxygen content of the starting material, or in terms of coal analysis and classification, of increasing the rank of the fuel. Eqs. 30 and 31 are descriptive of two hypothetical examples to the point:

$$\text{R.OH.COOH} + 2H^+ + 2e^- = \text{HR.COOH} + H_2O; \quad \Delta F^0_{298} = -15550 \text{ cal.}; \quad E^0 = +0.34 \text{ volt} \qquad [30]$$

$$2\text{R.OH.COOH} + 2OH^- = \text{HOOC.R.R.COOH} + 2H_2O + O_2 + 2e^-; \quad \Delta F^0_{298} = +43800 \text{ cal.}; \quad E^0 = -0.87 \text{ volt} \qquad [31]$$

Selection of examples 30 and 31 is based on the consideration that the so-called humic acids—which are hydroxycarboxylic acids—are the main components of brown coals and lignites; i.e., coals of lowest rank. The arrangement of Eq. 31 is based on results

of lignin and humic acid research and further, on the consideration that hydrogen peroxide is a likely intermediate of anaerobic life. This is shown by the fact that microbial cells are equipped with the enzyme peroxidase, which accomplishes the decomposition of hydrogen peroxide according to Eq. 32.

$$2H_2O_2 = 2H_2O + O_2; \quad \Delta F^0_{298} = -50180 \text{ cal.} \qquad [32]$$

Eqs. 30 and 31 are computed on the basis of the observation that ΔF^0_{298} for the introduction of a phenolic hydroxyl group is −41,000 cal., and for the introduction of a carboxyl group, −90,000 cal. Finally, it should be emphasized that the presence of the carboxyl group is not essential to the working of reactions 30 and 31. Phenols are amenable to the same type of reaction. A reduction process of type 31 will be the more complete the more alkaline and the more anaerobic the medium.

It has been the predominant opinion that increase of rank follows a natural trend: peat, it was thought, would gradually transform into lignite, bituminous coal, and anthracite, in that order, merely by exposure to geological forces. We have obtained quantitative estimates of geological forces; and we have noted that the introduction of oxygen containing groupings is more in line with natural trends than their removal. Apparently, however, a complete understanding of the subject can be attained only by a consideration of redox potentials: in a given medium, carbonaceous material is exposed to either oxidation or reduction processes depending upon the prevalent redox potential.

Thermodynamic Theory of Coal Formation

As a result of this investigation, a theory of coal formation based upon weighted thermodynamic probabilities is presented:

1. In an accumulation of moist plant debris, hydrolytic processes are the most probable ones. Through hydrolysis, ester, saccharide, and peptide linkages are severed; the large structures having those linkages will collapse, the split products being removed by biological action, and the residual water-insoluble structures that contain predominantly carbon-carbon linkages will accumulate. Lignins, resins, fatty acids, waxes,* and small amounts of condensation products from sugars, amines, and phenols† will then form the bulk of the deposit, the ultimate composition of which will approach that of lignin; i.e., 60 to 65 per cent carbon, 5 to 6 per cent hydrogen, and 30 to 34 per cent oxygen.

2. If the accumulation of carbonaceous material occurs under aerobic conditions in a moderate climate with scanty animal life, the nitrogen content of the medium will become exhausted; microbial activities will be inhibited, and slightly oxidized transformation products of lignin accompanied by resins, waxes, nonhydrolyzed cellulose, will form the bulk of the final fuel. These conclusions are in line with the paleobotanic investigations of W. Gothan[11] and with the chemical investigations of W. Fuchs[8] on German brown coals.

3. If the accumulation of carbonaceous material takes place under water, anaerobic conditions will prevail. Gothan and others[24] conceive of the areas of coal formation as extensive sinking areas, flooded by water, with a subtropic climate and abundant life. Under such circumstances, the nitrogen supply of the medium was replenished by contributions from animal life and plankton, and microbial activities were maintained. Carbonaceous material that accumulated in this medium from vascular plants and plankton was deprived of its oxygen content to a degree determined mainly by the prevailing redox potential,

* Waxes though esters are not easily hydrolyzed in a biological medium because they are not easily wetted by water and show no tendency to give colloidal solutions.

† From protein building units such as tyrosine or tryptophane.[7]

the reducing action increasing from shallow to deeper layers.

4. The presence of hydroxyl and carboxyl groups in brown coals and lignites is evidence that oxidation processes were essential in transforming the original plant material into these fuels. The absence of those groupings in bituminous coals and anthracites and their lower oxygen content indicate the importance of reduction processes in the formation of higher rank fuels. While lignites may have been reduced to give bituminous coals and anthracites, the simpler way is obviously that the original plant material underwent reduction without passing through a lignite stage.

5. Orogenic pressure aided in the compaction of coal up to values of 1000 lb. per sq. in. in bituminous coals, 4000 lb. per sq. in. in anthracites. Higher pressures tended to make the coals more friable.

6. As a rule, coal formation processes went to completion at temperatures well below 200°C., i.e. in a temperature range with little chance for thermally induced exothermic reactions, and little occasion for endothermic reactions. Much higher temperatures in coal deposits were attained by the influx of heat from volcanic sources. The sequence of exothermic and endothermic reactions realized with rising temperature resulted in a local beneficiation of the fuel, and in close proximity to the intrusion gave rise to materials resembling coke.

The relation of the thermodynamic theory to other theories of coal formation is briefly stated:

1. The lignin theory of coal states that in a deposit of plant debris the cellulose gradually disappears by microbial action while lignin, deprived of acetyl and methoxyl groups, accumulates as mother substance of the so-called humus coals. This theory, though oversimplifying more complex phenomena, works well for brown coals but fails to account for the contributions of nonvascular plant material and the change of rank in bituminous coals and anthracites.

2. The cellulose theory of coal denies the importance of the biological factor, at least in the formation of bituminous coals and anthracites, and explains formation of coal by the action of heat and pressure upon cellulose through long periods of time. This theory has been rejected previously on grounds of field evidence and is now shown to be improbable because of thermodynamic reasons.

3. The David White theory combines the essential features of the two preceding theories, by the assumption of two stages in the transformation of coal: the biochemical or peat-forming stage characterized by biological action, and the dynamochemical or coalification stage, during which the material was compacted and altered to various degrees both physically and chemically by the slowly increasing pressure of overlying sediments and side thrusts due to orogeny. It has been shown in the present analysis that pressure cannot possibly have the effects ascribed to it. Apart from volcanic intrusions, this is true also for the effects of heat, or heat and compression.

4. The McKenzie-Taylor[20] theory postulates that brown coals were formed aerobically in a slightly acid medium under a layer of calcium aluminum clay, while bituminous coals were formed anaerobically in a slightly alkaline medium under a layer of sodium aluminum clay. This theory contains elements of truth but postulates too narrow a mechanism and seems unable to account for the wide variety of rank in the coal fields. The thermodynamic theory is not restricted to a particular mechanism and supplies this explanation.

5. In studying the formation of coal, Terres[27] has emphasized the participation of proteins, and Stadnikoff,[26] of diolefinic and polyolefinic acids. The work of these investigators is compatible, and is also consistent with the previous work by

Fuchs[10] and with the results of the present investigation.

Summary

A thermodynamic analysis of the factors involved in the formation of coal and specifically in the change of rank of coal has been presented. Two stages of coal formation are recognized: the biological stage, characterized by hydrolytic processes and accumulation of more resistant material, and the biodynamic stage, characterized by oxidation-reduction phenomena in the medium.

Acknowledgment

Acknowledgment is made to Professors A. W. Gauger, D. R. Mitchell and R. F. Nielsen, of The Pennsylvania State College, and to Dr. T. Stadnichenko of the U. S. Geological Survey, Washington, D. C., for valuable criticism and suggestions.

References

1. Bridgman: A Condensed Collection of Thermodynamic Formulas, 26, Harvard University Press, 1925.
2. Bruins and Czarnecki: *Ind. and Eng. Chem.* (1941) **33**, 201.
3. Campbell: *Econ. Geol.* (1930) **25**, 680.
4. Fieldner and Selvig: Symposium on Industrial Fuels, 5, 9. Philadelphia, 1936.
5. Fischer: Bahr and Sustman: *Brennstoff Chem.* (1930) **11**, 1.
6. Francis: *Ind. and Eng. Chem.* (1928) **20**, 277.
7. Fuchs: Die Chemie der Kohle. Berlin, 1931.
8. Fuchs: *Brennstoff Chem.* (1930) **11**, 205.
9. Fuchs: *Braunkohle* (1933) **32**, 311.
10. Fuchs: *Fuel in Science and Practice* (1935) **14**, 168, 211.
11. Gothan: *Braunkohle* (1931) **30**, 227.
12. Gutenberg: Physics of the Earth, VII—Internal Constitution of the Earth. New York, London, 1939.
13. Heck: *Bull.* Geol. Soc. Amer. (1938) **49**, 1884.
14. Hewitt: Oxidation-Reduction Potentials in Bacteriology and Biochemistry. London, 1937.
15. Hoffman: *Angew. Chem.* (1936) **49**, 740; Beiheft No. 24 *Ztsch. V. D. Chem.*
16. Jeffreys: The Earth, Cambridge, 1929.
17. Jordan and Falk: The Newer Knowledge of Bacteriology. Chicago, 1928.
18. Latimer and Hildebrand: Reference Book of Inorganic Chemistry, 482. New York, 1940
19. Lewis and Randall: Thermodynamics. New York, London, 1923.
20. McKenzie-Taylor: *Fuel in Science and Practice* (1928) **7**, 230; *Nature* (1928) **121**, 789.
21. van Orstrand: See Ref. 12, 147.
22. Parks and Huffman: The Free Energies of Some Organic Compounds. Amer. Chem. Soc. *Monograph*, 1932.
23. Petrascheck, see Entstehung, Verdelung und Verwertung der Kohle, 1ff. Berlin, 1930.
24. Schuchert and Dunbar: A Textbook of Geology, Ed. 3, pt. II, 241ff. New York, 1933.
25. Stadnichenko: *Econ. Geol.* (1934) **29**, 511.
26. Stadnikoff: Die Entstehung von Kohle und Erdöl. Stuttgart, 1930.
27. Terres et al.: *Angew. Chem.* (1935) **48**, 160; Beiheft No. 13 *Ztsch.* V. D. Chem.
28. G. Thiessen: *Trans.* Ill. State Acad. Sci. (1936) **28**, 184.
29. R. Thiessen: What is Coal? Meeting Fuel Eng. Appalachian Coals Inc., Cincinnati, 1937.
30. Treibs: *Angew. Chem.* (1936) **49**, 682.
31. Trifonow and Toschew: *Brennstoff Chem.* (1940) **21**, 85.
32 Waksman: Principles of Soil Microbiology. Baltimore, 1932.

A Continuously Operating Laboratory Coal Pulverizer That Measures Net Power

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(Birmingham Meeting, November 1940)

Data concerning the actual net energy required for pulverizing coal are lacking from the literature on coal pulverization. Power data given in the literature concern gross power and frequently include the power expended on the air blast or product-conveyor system, as well as power lost through friction in the pulverizer itself. Consequently, such data supply no information regarding the net energy consumed in pulverizing or the relative grinding resistance of various coals.

In the study of coal pulverizing much work has been done on what is known as "grindability-index numbers," but grindability tests have seldom considered the energy input (that is, the "cause" part of the system "cause and effect"). There are many records of the effect, but even these have questionable value because the "grindability" tests did not make a pulverized product by a method comparable with commercial pulverizers. In many of the tests, only a selected part of the coal was ground, and the remainder continued to be an unknown factor. Furthermore, none of the grindability tests provide the balanced circulating load common to all modern commercial pulverizing.

In order to obtain better understanding of the fundamental principles of coal pulverizing, the Bureau of Mines is investigating the "cause" phase; that is, the net energy input or net power excluding friction between the moving parts of the machine.

As a first step in the investigation, it was necessary to design and construct a laboratory machine in which no friction-loss corrections are involved, which could grind and finish a minus-¾-inch feed, which could be emptied rapidly and completely, which could operate with a continuous feed and finish in an air separator; and, most essential of all, which would be equipped with a measuring device for recording only that portion of the total power consumed in actual pulverizing, hereafter referred to as "net energy input" or "net horsepower-hours." The channel-roller pulverizer hereinafter described is believed to meet these requirements.

History of the Machine

The work started when Ralph W. Smith, engineer, St. Genevieve Lime Co., St. Genevieve, Mo., was appointed one of the consultants of the Bureau of Mines, United States Department of the Interior, in cooperation with the University of Alabama, to develop a machine for testing the abrasiveness of coal. Smith had already developed a machine for testing the abrasiveness of dentifrices,[1] and it was thought the same principles might be applied to coal. Smith's machine had a revolving disk with an annular channel in which the sample was placed. Riding in the channel was a small block or runner of selected soft metal held

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[1] R. W. Smith: Machine for Testing Dentifrice Abrasion. *Ind. and Eng. Chem.*, Anal. Ed. (1940) **12**, 419–423.

stationary by a specimen holder. The amount of wear on the metal block indicated the abrasiveness of the sample.

The first tests in the Bureau laboratory concerned the abrasiveness of coal; but while these were in progress, the idea developed that the net energy input could be calculated by measuring the tangential force against the runner and the distance turned by the channel; the power required to turn the disk would not enter into the calculations. Accordingly, thoughts turned to grinding instead of abrasion, and the runner of the original abrasion machine was replaced by a roller, held in position by radial arms, at the outer end of which the pull was measured. The measuring of the distance turned off by the channel was simple, but the recording of the variable tangential forces required long study and experimentation.

The first machine had an 11-in. channeled disk and was limited to a minus 20-mesh feed. The force recorder was based upon a balanced pendulum and rack. Experience with the machine indicated that the principles could be applied on a much larger scale, and so the larger machine described herein was designed and constructed. In this machine the grinding is continuous, and finishing is by air separator, adjusted to give a product comparable with that of the commercial plant.

The authors take pleasure in acknowledging their indebtedness to others who have worked on the problem at the Southern Experiment Station, Bureau of Mines, Tuscaloosa, Ala. The first machine was built and tested by Henry Landson, Assistant Chemical Engineer (deceased December 1938). Much credit is due John Dasher, Junior Chemical Engineer, who continued with the development until his transfer to the College Park Station, Maryland, in April 1940. He was assisted for a short time by William Sheppard, Junior Mining Engineer.* M. F. Williams, Jr., Junior Physical Chemist, assisted with the development and testing of the large machine and G. T. Adams, Senior General Mechanic, assisted with the development and construction of both machines. The electrostatic precipitator was designed from information supplied by O. C. Ralston, Chief Engineer, Nonmetals Division, Bureau of Mines, College Park, Maryland.

The Channel-roller Pulverizer

In operation of the channel-roller pulverizer, coal is fed to and carried in the channel of a revolving horizontal disk, designated in Fig. 1 by the letter *C*. A free-floating roller *R*, held in a vertical position by means of radial arms (*T*) rides on the coal in the channel. The radial-arm suspension of the roller permits free movement in the vertical plane. The roller-radial-arm assembly is pivoted on a shaft *S* supported by ball bearings in the tubular shaft of the channeled disk. This arrangement permits the roller to swing freely in the horizontal plane through an arc of the same radius as that of the channel. In operation, the roller is restricted in its arc of horizontal swing by the link *L*, which is attached to the recorder. The channel must be perfectly level in the direction of the pull of the roller.

The roller comprises a solid cast-iron core on which has been shrunk a hardened steel tire. The channeled disk is mild steel. These materials have proved entirely satisfactory for pulverizing coal, but if the machine were to be used on more abrasive materials a hardened steel liner should be attached to the bottom of the channel, not only to reduce wear but also to offer a harder grinding surface.

Tangential-force Recorder

The force recorder consists of an arm held in a vertical position by opposed balancing coil springs and supported by a ball bearing at the bottom. The arm is free to swing in the plane in which the springs are mounted, but any motion is opposed by the pull of

* Resigned to enter Army Air Corps June 30, 1939.

the springs. Only the top of the arm is visible in the photographs, the greater portion being hidden by the chart. Movement of the arm in a plane at right angles to the ball-and-socket attachment at the left end of link *L*. Thus any force that produces a radial movement of the roller assembly is transmitted through link *L* to the pivoted

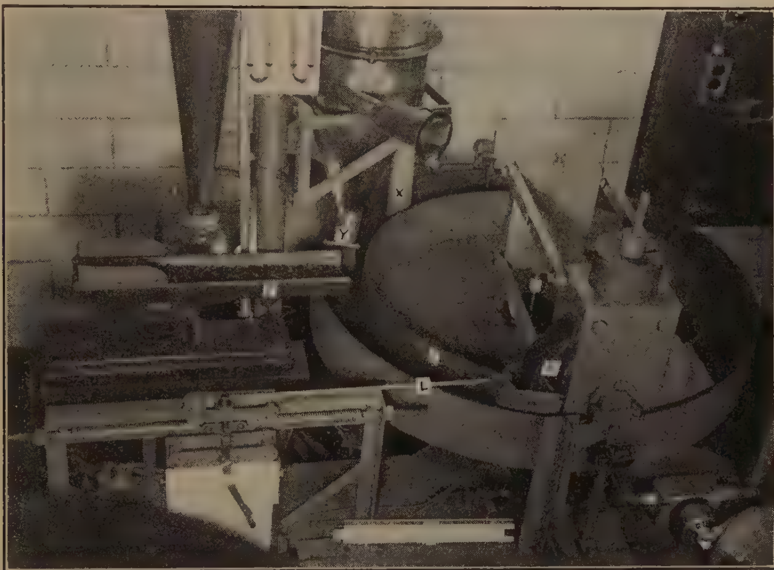

FIG. 1.—CLOSE-UP OF CHANNEL-ROLLER PULVERIZER.

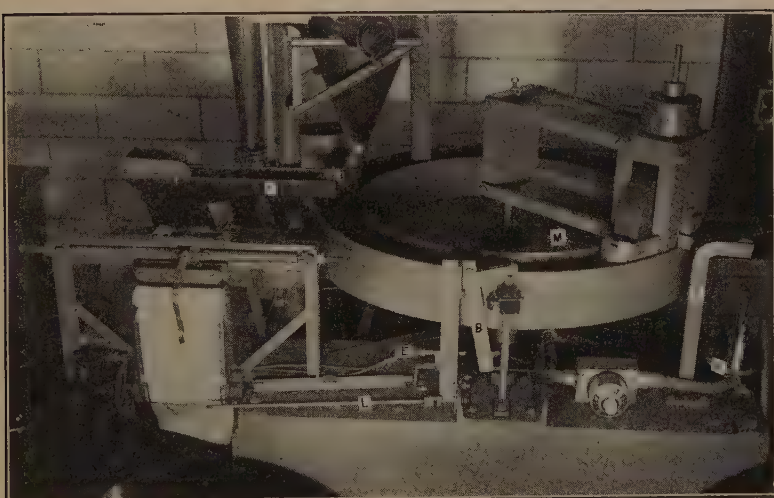

FIG. 2.—METHOD OF CALIBRATING THE FORCE RECORDER.

springs is prevented by a ball bearing mounted on the top of the arm and running between two steel guide bars. This bearing is shown in Fig. 1 immediately beneath the arm. The arm, in turn, swings in the direction of application of the force a distance sufficient to produce in the coil springs a tension that exactly balances the force on

the roller. Any movement of the arm is magnified and recorded on a continuous paper chart through the pivoted pen carrier shown on the chart in Fig. 2. The arm

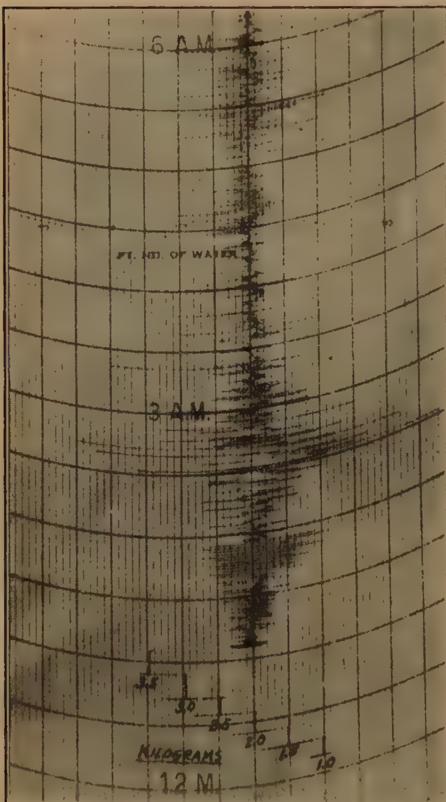

FIG. 3.—SECTION OF RECORDER CHART.

is connected with the pen carrier by means of a safety-pin linkage, which provides a uniform pen movement for uniform movement of the arm, even though the radii of attachment of the two have a ratio of 1 to 7½. The continuous paper chart beneath the pen is driven, by a small reduction-gear electric motor, at the rate of 1¼ in. per minute.

To calibrate the recorder, the connecting link is removed and a fishing line supporting a definite known weight is attached to the recorder arm at the ball-and-socket joint. This is illustrated in Fig. 2, which shows the roller swung to one side and the fishing line *M* supporting the weights *W* in position for calibrating. The post *P* carries a ball-bearing roller, which changes the direction of pull of the weights from vertical to horizontal. Different weights produce definite deflections of the springs and the pen records these deflections on the chart. Thus the chart is calibrated throughout the range of tangential forces met in operation. Fig. 3 is a reproduction of a section of a recorder chart. At the bottom of the figure the chart calibration is shown, and the wavy line in the upper two thirds of the figure is the record of the tangential forces on the roller during an actual continuous pulverizing test.

Four sources of error in the recorded tangential force must be avoided:

1. An inclined channeled disk. Any tilt to the disk will cause the roller to run uphill or downhill, and result in the recorded tangential force being greater or less than the true force.

2. Friction in the bearing of shaft *S*, Fig. 1, which serves as a pivot for the roller-toggle-arm assembly. Precision-ground ball bearings were used to support shaft *S*, and attempts to measure the bearing friction have proved beyond doubt that the bearing friction in this machine is negligible.

3. Friction in the recorder. Here again precision-ground ball bearings were used throughout, and the friction is insignificant.

4. Improper alignment of link *L* and roller *R*. To measure the true tangential force acting on the roller the measurement must be made on a line parallel to the side of the roller and through the center of the face. Since any horizontal movement of the roller is radial, link *L* does not always parallel the side of the roller. However, the length of the link is adjusted so as to maintain alignment of the link and roller at the average position occupied by the roller in the actual pulverizing tests. The springs of the force recorder limit the radial move-

ment of the roller, so that the maximum error due to angularity of pull is 0.1 per cent.

Automatic Feed Control

An automatically controlled feed is provided by the electrically operated Jeffrey vibrating feeder *F* and switch *BE* shown in Fig. 2. When the roller *R* is in operating position, a ball-bearing roller at the end of the offset leg of arm *B* presses against the bottom of a steel block mounted on the roller saddle. As the thickness of the bed of coal in the channel decreases because of the removal of finished product, the roller *R* rides in a lower position and moves the arm *B* to the left. On the lower end of arm *B* is an electrical contact, which engages another contact on the micrometer *E*. When the contacts on *B* and *E* close, feeder *F* delivers coal to the channel until enough has been added to raise the roller and thus break the contact. The micrometer adjustment of the switch permits any desired depth of bed to be carried in the channel. Usually the micrometer is set to maintain a bed of about 1 lb. in the channel, providing a layer about ¼ in. thick. The feed rate is thus controlled automatically by the rate of removal of finished product, and has ranged from 2¼ lb. per hour for a very hard bituminous coal to 10 lb. per hour for a very soft coal. A heavier weight on the roller increases the grinding rate.

Cyclone and Separator

A continuous discharge of pulverized product is accomplished with the cyclone collector and air separator shown in Fig. 4. In the upper right of Fig. 4 is shown the exhauster, which is connected to the electrostatic precipitator (upper left). The dust inlet to the precipitator is connected to the exhaust of the cyclone, and the cyclone inlet is connected to the exhaust of the air separator. The air separator and connection to the cyclone are partly hidden by the manometer panel. The air-separator inlet and the tailing spout dip into the channel and are shown more clearly in Fig. 1. In operation, suction applied at the air-separator inlet *X* draws most of the

Fig. 4.—Channel-roller pulverizer with accessories, in operating trim.

material of finished and near-finished size into the separator. The separator usually is adjusted to make a split at about 150 mesh, though the top limiting size of the separation can be varied through a wide range. The two manometers shown in Fig. 1 measure the pressure drop across the cyclone and separator and enable the operator to maintain a constant suction. Any fluctuation in suction in the separator will cause a fluctuation in the particle size and output of the pulverized product. Consequently, uniform suction should be maintained throughout a test. Oversize material from the separator is returned directly and continuously to the channel through the

tailing spout provided with a flap gate *Y*. The undersize is carried from the separator into the cyclone, from which it is discharged into a glass jar. On the coals tested so far, the cyclone product has constituted 98.7 to 99.1 per cent by weight of the total feed. The air separator is the standard laboratory-size separator manufactured by Federal Classifier Systems, Inc. The cyclone was designed by the authors.

FIG. 5.—CYCLONE ESCAPE DUST COLLECTED IN ELECTROSTATIC PRECIPITATOR.
1 division = 4.2 microns.

ELECTROSTATIC PRECIPITATOR

The loss of about one per cent of coal dust in the cyclone exhaust could not be overlooked, and an electrostatic precipitator formed of two concentric cylinders with multiple ionizing wires in the space between was designed to complete the recovery. Use of the apparatus prevents loss of any of the sample as dust.

Electrostatic precipitation of the cyclone escape dust is practiced only in connection with tests designed for the study of fundamentals wherein particle size and surface measurements are to be made. Tests of this type are, as yet, considered impractical for most industrial needs. Since the electrostatic precipitator may be omitted for most work, details of the precipitator will not be discussed here. Suffice it to say that the effective length of the precipitator column is 4 ft. and the electrode gap is 1 in. The outer cylinder is 8 in. in diameter and the inner cylinder 4 in. The cylinders are of polished brass; 12 tungsten wires each 0.003 in. in diameter are suspended in the space between. The power supply is 7000 volts of pulsating direct current furnished by a neon-tube transformer and vacuum-tube rectifier. The precipitator collects virtually 100 per cent of the cyclone escape dust. Fig. 5 is a photomicrograph of a product from the precipitator and shows that all the particles that escaped collection in the cyclone were less than 3 microns in diameter.

OPERATION AND TEST PROCEDURE

Fig. 4 shows the machine in operating trim. The roller is linked to the recorder arm and bears on the layer of coal in the bottom of the channel. The direction of rotation of the disk is counterclockwise. Under these conditions, and with the disk revolving, the roller tends to swing in the direction of rotation of the disk. As mentioned before, the distance through which the roller can swing is restricted by the recorder springs, and the tendency to swing (or tangential force) is counterbalanced by the springs and recorded in terms of pounds or kilograms. From the circumference of the channel and the number of revolutions can be ascertained the distance through which the tangential force is applied. The product of force-times-distance (or work) may be expressed as foot-pounds, horsepower-hours, or kilowatt-hours. The result,

combined with the weight of pulverized coal, may be expressed as horsepower-hours per ton of finished product. The formula for converting the laboratory data for this particular pulverizer is:

$$\text{Horsepower-hours per ton of pulverized product} = \frac{\text{Tangential force (kg.)} \times \text{revolutions} \times 7.07}{\text{Grams finished}}$$

The factor 7.07 applies only to this particular machine and considers the circumference of the channeled disk in addition to the factors for converting metric units to English units and foot-pounds to horsepower-hours. The weight of the roller does not enter directly into the calculation but is reflected in the tangential force. The greater the roller weight, the greater will be the tangential force. The machine at present is operated at a disk speed of 19.4 r.p.m. (which corresponds to a travel of 136 ft. per min), and with a total roller weight of 35 to 40 kg. Under these conditions, the average tangential force on most coals is about 2 kilograms.

The investigators do not claim that the principle of operation of the channel-roller pulverizer is the most efficient or is more efficient than that of any other type of comminuting machine, but do claim that the net energy required to pulverize a coal need not exceed that consumed by the channel-roller pulverizer on the same coal. We regard the energy expended in overcoming interparticle friction in the channel, as well as that expended in elastic deformation without rupture, as integral parts of the energy required for grinding. In other words, the amount of power measured in any machine must be recognized as pertaining to the particular combination of attrition and impact common to that machine.

Preparation of Sample

In the early tests with the pulverizer, variations in the power required to pulverize a coal were traced to variations in moisture content. Consequently, all tests are now made on samples that have been thoroughly dried at 110°C. The energy input for consecutive 1-lb. charges is likely to fluctuate unless every charge has the same size composition. Ordinary quartering or riffling methods are not satisfactory for cutting 1-lb. charges from the bulk sample of minus ¾-in. coal. To obtain more uniform charges, the following method has been adopted. It is reasonably accurate and not too time-consuming. A 10 to 15-lb. sample of the coal to be tested is screened on 0.525-in., 3-mesh, and 8-mesh sieves and the increments are weighed. From the size distribution thus determined the weight of these size increments necessary to make up a total sample of 1 lb. is calculated. The calculated amount of each of the four sizes (−0.742 + 0.525 in., −0.525 + 3 m., −3 m. + 8 m., and −8 m.) is weighed out, and the increments are recombined. Thus, ten to fifteen 1-lb. charges are obtained. In making a test, about 1 lb. of the minus ¾-in. dried coal is put in the channel, and, with the roller lifted, the channel drive motor is started. Three small rakes dipping into the channel in close proximity to the air-separator inlet tube distribute and level the coal. During the dozen or so revolutions required to distribute the feed in the channel, the feeder is filled with dried coal. The exhauster is then turned on, and simultaneously the roller is lowered onto the coal, the recorder-chart drive is started, and the revolution counter is engaged with the channeled disk. After the pulverizer is started, the only attention required is to see that the feeder never becomes completely empty and that the glass jar in which the cyclone product is collected is replaced as soon as it is full. Although the pulverizer is operated as a continuous and not a batch testing machine, the laboratory practice is to record the rate of grinding for each increment of about 1 lb. of pulver-

ized coal produced. The pulverized product from the cyclone is collected in 1-qt. glass jars, and at the instant each jar of product is removed from the cyclone the number of revolutions registered on the counter is recorded. From the weight of product removed and the number of revolutions, the operator, during the test, is informed as to the rate of grinding. When the weight per revolution remains constant for at least four successive jars of product, it is considered that the separator return and channel load have become constant, so that the feed and the pulverized product are in balance. The test may then be stopped. In calculating the energy input, the tangential forces recorded before the point at which a balanced load was achieved are disregarded. The time required for the actual pulverizing test is about 2½ hr.; this does not include the time necessary for preparing and drying the feed sample.

Danger in Short-cut Testing

The performance of the channel-roller pulverizer is analogous to an air-swept ball mill followed by an air separator and cyclone. The construction of the pulverizer, however, has the advantage of exposing the channel load to inspection whereas, because of the inaccessibility of the ball mill, little attention is given to the nature of the product built up in the mill.

In modern pulverizing, hard particles in the feed accumulate in the circuit. When they have accumulated so that they are ground out as rapidly as the same constituents enter as part of the new feed, then, and only then, is the circuit balanced, and only then can the power, the amount, and the nature of the finished material be appraised.

The time required for a balance depends on the nature of the feed. If it is homogeneous, the balance is reached quickly, but if hard streaks are present, as in most coals, the time required can be found only by trial. A coal may contain a little as 1 per cent of hard pyrite, an amount so small that it might be overlooked by those who are working in haste; however, this pyrite might build up in the channel and circulating load to the extent that it would occupy much of the space allotted to virgin material. It might be, and in some instances has been, a vicious diluent.

Dilution by resistant material has so much influence that it requires more consideration. This evidence is offered in Table 1, which shows the results of continuous pulverizing of ¾-in. washed coal in closed circuit with an air separator. The coal had a tenor of only 9 per cent ash, but the material remaining in the channel of the pulverizer at the end of the run was 20.3 per cent ash; several of the coarser sizes had the staggering amount of more than one third ash. In ordinary milling, such material would have been housed in the mill and remained unobserved. Obviously any results obtained without allowing for this material will be in error. Any substance that is not homogeneous will show these characteristics. The resistant material is not always high in ash; some low-ash coals are harder than the associated gangue minerals. Any investigator working without respect for heterogeneity that causes hard material to build up in the mill or pass out to be returned is liable to be misled. In the channel-roller pulverizer the channel load and the separator return may be sampled quickly and accurately. The ease with which the products may be sampled and the fact that the cycle of operation duplicates accepted commercial practice are features overshadowed in importance only by the ultimate object—the measurement of net energy consumed in pulverizing rather than the abstract and relative "grindability indexes."

Application

Although it is probable that the channel-roller pulverizer will prove applicable to dry fine grinding of many materials, only coal is considered here. The application to

coal is twofold: (1) tests for industrial control and (2) tests of a more theoretical or fundamental nature. These will be considered in turn.

TABLE 1.—*Ash Content of Channel Load in Pulverizing Washed Coal in Closed Circuit with Air Separator*

¾-INCH COAL, 9 PER CENT ASH

Channel Load

Size, Mesh	Weight, Per Cent	Ash, Per Cent
−0.371 in. +3	0.7	17.2
−3 +4	1.7	32.1
−4 +6	4.2	34.9
−6 +8	5.3	26.6
−8 +10	3.4	37.2
−10 +14	6.7	25.8
−14 +20	8.4	20.8
−20 +28	9.4	25.0
−28 +35	10.7	20.3
−35 +48	8.8	19.6
−48 +65	7.8	19.7
−65 +100	8.8	17.5
−100 +150	6.6	14.1
−150	17.5	10.0
Cumulative	100.0	20.3

Tests for Industrial Control

Table 2 contains the results of three pulverizing tests of an industrial type. Plant operators frequently judge pulverizer performance by the percentage of the product finer than 200 mesh. However, an increasing number of operators have accepted the idea that the percentage passing a 200-mesh sieve is less important than that no more than 1 per cent remain on a 48-mesh sieve. In view of this, both the percentage of plus 48-mesh and percentage of minus 200-mesh are recorded in Table 2. The data for coals A and B (both washed coals) show that the extreme fluctuation in power for the last four charges is less than 5 per cent, so it may safely be assumed that the circuit was balanced and the net energy consumed in pulverizing these coals to 99.8 per cent minus 48-mesh was the average of the last four charges, or 6.6 and 9.5 hp-hr. per ton. The data for coal C (raw coal) are included to demonstrate further the progressive building up of relatively hard material in the pulverizer. This shows that the first charge required only 5.2 hp-hr. per ton, whereas after 82 min. this value had increased to 7.0 hp-hr. per ton. Moreover, this test demonstrates that the raw coal arrives at a balanced load more slowly than does the more homogeneous washed coal. Furthermore, it should be apparent that the test of coal C is unfinished. The net horsepower-hours per ton has pro-

TABLE 2.—*Tests for Industrial Control*

Description	Grams per Revolution	Average Tangential Force, Kg.	Net Hp-hr. per Ton	Pulverized Product, Wt. Per Cent	
				+48-Mesh	−200-Mesh
Coal A (washed); total pulverizing time, 130 min.	2.23	1.76	5.6		
	1.91	1.75	6.5		
	1.93	1.73	6.3		
	1.95	1.78	6.5		
	2.05	1.77	6.1		
	1.95	1.75	6.4		
	1.92	1.76	6.5		
	1.90	1.78	6.6		
	1.89	1.79	6.7		
	1.90	1.79	6.6	0.2	61.9
Coal B (washed); total pulverizing time, 98 min.	1.32	1.64	8.8		
	1.23	1.64	9.4		
	1.25	1.67	9.4		
	1.24	1.67	9.5		
	1.21	1.68	9.8	0.2	46.7
Coal C (raw); total pulverizing time, 82 min.	2.30	1.70	5.2		
	2.29	1.81	5.7		
	2.03	1.80	6.2		
	2.03	1.80	6.3		
	1.99	1.84	6.5		
	1.92	1.87	6.9		
	1.89	1.88	7.0	0.7	39.6

gressively increased, and there is no evidence that 7.0 hp-hr. per ton (the last value shown) is the correct value for this coal. Let it be emphasized further that a pulverizing test should continue until the extreme variation in net horsepower-hours per ton for at least four consecutive charges does not exceed 5 per cent of the average of the four charges. This 5 per cent limitation on extreme variation in energy input may appear to be arbitrary. This value was tentatively adopted for the industrial type to test, to ensure that the test be carried to completion and that the average energy value be sufficiently accurate for industrial purposes, as well as to avoid prolonged testing in an attempt to achieve inordinate precision. Laboratory tests have shown that, in order to comply with this limitation, some (but not extreme) care must be exercised in the preparation of the feed sample.

TABLE 3.—*Laboratory Tests for Study of Fundamentals of Coal Pulverization*

A Feed to Pulverizer	B Charge No.	C Net Power Input, Hp.	D Output, Tons per Hr.	E Tons per Hp-hr. (D/C)	F[a] Units of Surface per Unit Weight	G Surface Tons per Hp-hr. (E × F)
Jig feed, minus ¾-in.; ash, 17.4 per cent	6	0.0181	0.00173	0.0955	3,140	300
	7	0.0181	0.00183	0.1010	2,800	283
	8	0.0178	0.00165	0.0927	3,210	298
	9	0.0179	0.00182	0.1016	2,830	288
Average		0.0180	0.00176	0.0978	3,000	293
Washed coal, minus ¾ in.; ash, 8.4 per cent.	6	0.0164	0.00140	0.0854	3,000	256
	7	0.0164	0.00142	0.0865	3,000	260
	8	0.0167	0.00145	0.0868	3,020	262
	9	0.0164	0.00145	0.0884	3,080	272
Average		0.0165	0.00143	0.0867	3,030	263
Jig refuse, minus ¾-in.; ash, 67.4 per cent.	6	0.0182	0.00194	0.1065	4,250	452
	7	0.0182	0.00186	0.1020	3,750	383
	8	0.0182	0.00195	0.1070	3,790	405
	9	0.0182	0.00191	0.1050	3,830	402
Average		0.0182	0.00192	0.1055	3,910	412

[a] Surface is total surface measured by the air-permeability method.

Tests for Fundamentals

The data in Table 3 are the results of tests designed for the study of certain fundamentals of coal pulverization. Inasmuch as the final results are expressed in terms of surface tons per horsepower-hour, surface must be measured, and in order that these measurements be accurate the electrostatic precipitator must be used to collect the dust escaping from the cyclone.

The accuracy of the surface values recorded in Table 3 is somewhat questionable. The problem of surface measurement has required considerable research, and at this time the air-permeability method is believed to be the most suitable from the standpoint of speed, range, and accuracy. Improvements in the method have been made, and research is continuing in an endeavor to achieve an accuracy commensurate with that of the power measurements. The method and an improved apparatus are described in a paper being prepared for publication (Schuhmann et al.: The Air-permeameter).

The data of Table 3 illustrate a condition formerly mentioned; that is, some coals are harder than the associated gangue. Here is a jig refuse considerably softer (412 surface tons per horsepower-hour) than the coal from which it was washed (263 surface tons per horsepower-hour). Table 3 is included primarily to show the type of information required for a study of the fundamentals of coal pulverization.

Cost of Materials

Pulverizer and force recorder	$238.47
Exhauster, air separator, cyclone and feeder	273.00
Electrostatic precipitator	68.00
Total	$579.47

Summary

The construction and operation of a laboratory coal pulverizer capable of continuous operation on ¾-in. feed in closed circuit with an air separator have been described. The distinguishing characteristics of the pulverizer are:

1. The pulverizer is so constructed and equipped that a measure of net energy (that is, the portion of the total energy input that is expended on the coal itself) is possible and accomplished by means of the force recorder described. Net energy is expressed as "horsepower-hours per ton."

2. The various steps in the operating cycle simulate accepted commercial pulverizer practice in automatic balance between the rate of new feed and rate of discharge of finished product, pulverization by means of heavy roller, air separation of the finished pulverized product, and an oversize product automatically returned for further size reduction.

3. The laboratory machine produces a pulverized coal that closely approximates commercially pulverized coal in limiting size and size distribution.

4. The separator return and channel load can be readily sampled to determine their characteristics.

5. An ever-present danger in short-cut testing for grinding rate is discussed.

6. Both tests for industrial control and tests for fundamentals are illustrated.

The By-product Coke Oven in Defense and Industry

By C. J. Ramsburg*

(New York Meeting, February 1942)

The construction and operation of by-product coke-oven plants in America are essential to strong national defense and of the greatest importance to many widely diversified undertakings as well as to safety and health.

Carbonization Process

Coal used under a boiler or in a locomotive is consumed by combustion. Coal used in a by-product oven is distilled or carbonized, meaning that heat is applied to the coal, out of contact with air, causing a distillation that removes the volatile matter in the coal in the form of gas and vapor, leaving coke as a final product. From the gas and vapor distilled from the coal, many valuable substances can be secured, known as by-products. Hence the name "by-product oven."

The simplest example of this principle can be shown by means of an ordinary clay pipe. If powdered coal is placed in the bowl of the pipe, the opening of the bowl is plugged with clay, and the bowl placed in a flame, the bowl will become hot, transferring the heat through the wall of the bowl to the coal within. When this coal attains a temperature of about 750°F., gas will begin to pour out of the pipe stem, and this gas may be lighted. A black liquid also will exude from the stem of the pipe. This is tar mixed with certain other products. What remains in the bowl after distillation is coke.

A coke oven corresponds to the bowl of the pipe. A modern oven of the latest type is a refractory brick chamber approximately 42 ft. long and 14 ft. high, averaging 18 in. in width. The oven is filled with coal through closable charging holes in the oven top. Each end of the oven consists entirely of a self-sealing door, the removal of which permits the finished coke to be discharged by means of a power-operated pusher machine. The ovens are built in batteries, and the heating is effected by gas burned in vertical flues in the oven walls, each set of flues heating the walls of two adjacent ovens. Such ovens carbonize a charge of approximately 18 tons of coal in 18 hr., or have a daily throughput of 24 tons per oven, producing about 17 tons of coke. In times of need, the output may be increased by raising the temperature of flues, thus shortening the coking time.

There is one great essential difference in principle between distillation in a clay pipe and in a coke oven. In the 18-in. oven, the distillation proceeds from two parallel walls held at a high temperature (over 2000°F.) and instead of all the coal being heated at once, the distillation is a progressive one creeping inward. The average travel of this heating is ½ in. per hour from each vertical wall, and as the volatile products are formed progressively they must push away from the interior toward the wall through the previously formed hot coke, then up the walls and over the top of the coal charge, finally escaping through the outlet pipe. This results in a cracking of the components into many different and valuable substances not formed when the carbonizing temperature is low.

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Coal Mining vs. By-product Ovens

As to their importance in the fuel economy of the nation, the by-product coke ovens in operation and building will shortly have a capacity under normal operating conditions to carbonize more than 90,000,000 tons of coal per annum, producing approximately 63,000,000 tons of furnace, industrial, and domestic coke. From the coal industry's standpoint, this means an outlet for more than one-sixth of the total coal mined. It can also be put in a striking way by the statement that every minute, night and day, three cars of coal are poured into by-product coke ovens, with the production of over 1,000,000 cu. ft. of coke-oven gas per minute for use in steel manufacture, in industrial plants of various types, and in domestic appliances. In 1942, there will be approximately 13,500 ovens in operation, distributed widely over the United States in 90 plants, varying in size from 15 ovens with a capacity of 44,000 tons per year to a plant of the Steel Corporation, the largest in the world, consisting of 1472 ovens with a capacity of 11,500,000 tons per year. More than 1000 ovens are under construction at present, with a capacity of approximately 9,000,000 tons of coal per annum.

Iron and Steel vs. By-product Ovens

The steel plant is interested in three coke-oven products used directly in manufacture of pig iron and steel—coke, gas, and to a smaller extent, fuel tar. The by-product oven has become an integral and necessary part of a modern steel-plant layout.

Two important results from the substitution of the by-product oven for the old beehive type have been secured in the operation of the blast furnace; i.e., operation on by-product coke produces more iron per day with greater uniformity of quality and with considerably less coke per ton of iron.

Because by-product coke plants are built at or near blast-furnace plants, coal from outside sources can be delivered to them from select-quality mines and can be blended to suit the operator. As the investment in that type of coke plant is so large and important, there must be provided a standby quantity of coal in order that interruption in coke supply may not be engendered by strikes and traffic interruptions. During the soft-coal strike in April 1941, there was little interruption in coke production at such plants, although furnaces using beehive coke were almost immediately closed down. Beehive coke ovens are peak-load operations and cannot afford such coal-storage facilities.

Many direct and many contingent advantages lie in the operation of a by-product coke plant in conjunction with a steel plant. To summarize these briefly:

1. As compared with beehive-oven operation, there is a saving in the amount of coal used in producing a ton of coke.

2. By-product coke is made from finely pulverized coal, thus distributing ash, sulphur and other impurities and producing a coke of uniform quality from day to day, making the operation of the blast furnace more efficient and increasing output of pig iron, with a smaller coke consumption.

3. The large amount of available gas produced adds to the economy and efficiency of the whole steel plant.

4. The credit for gas, tar, ammonia, light oil and other products pays substantially more than for fixed charges and operating expenses.

5. The magnitude of the investment makes it imperative that coal be held in storage in such quantity as to tide over mining and transportation interruptions.

Coal is so commonly used, and is such an everyday matter, that few stop to think what a wonderful material it is. Few know that coal can be dissolved almost com-

pletely in certain oils—that coal can exist in the form of a liquid, with a large part of the ash removed. Such a phenomenon is largely of academic interest because the process is too costly to be practical with present knowledge. Such a process may be developed in the future to produce an ash-free coke, though it is not possible now. This point is mentioned to illustrate the fact that in the distillation of coal a raw material quite as complex as petroleum is being used, capable of enormous development into many wonderful substances.

Value of the So-called By-products

When the gas passes out of the coke oven, it carries with it three major products: coal tar, light oils, ammonia; in addition, in smaller quantities, cyanogen and sulphur.

Coal Tar

The quantity of coal tar produced varies with the type of coal carbonized but the average amount produced in the United States is 8.6 gal. per ton, so that the total output if 90,000,000 tons of coal were being used would be close to 764,000,000 gal. What becomes of coal tar? At times the steel industry uses as fuel in open-hearth furnaces as much as half the amount it produces. This fuel is very desirable because it is low in sulphur and has a flame of great radiant heat, especially valuable in open-hearth steel manufacture. Several of the large steel plants make a practice of "topping" their tar before using it in open-hearth operations. This removes the lighter boiling fractions, which contain valuable tar acids, naphthalene and light oil. These in turn are sold to the tar plants for refining. The amount so used, however, is governed by economic conditions involving the comparative price of petroleum fuel oil and the price offered by the tar distiller. What the tar distiller can pay in turn is governed by the markets for his products. The tar-distilling plant has four major outlets: road tars, creosote oil, pitch, and naphthalene. All the tar, except for a very small percentage, goes into these materials. A misconception exists in the minds of many as to the nature and chemical composition of coal tar. The tendency to popularize science has found fertile soil in the field of organic chemistry. In their efforts to impress, writers have gone beyond the actual facts, with the result that today there exists around coal tar a webby fabrication, the structure of which can only be described by the phrase, "a rainbow of colors."

Much of this misconception arises from the fact that many people regard coke-oven light oil as a product of coal tar, when as a matter of fact it is obtained as a separate product and from an entirely different part of the coal-carbonization by-product recovery system. Coal tar condenses right at the beginning of the recovery system, whereas the light oil is virtually the last product to be removed. Light oil contains the benzol, toluol, xylol and the trimethyl and tetramethyl benzenes, such as mesitylene, pseudocumene, and isodurene. It also contains indene, coumarone and styrene—the resin-formers—as well as pyridine and its homologues, the picolines, lutidines, collidines, and possibly some of the parvolines. Thus it is that benzol and toluol, which together account for about 60 per cent of the total number of intermediates for dyes and other synthetic organic chemicals, are not obtained from coal tar. Consequently, it is not proper to refer to materials such as saccharine, acetanilid, aniline and T.N.T. as coal-tar products. It is true that the by-product recovery system is not 100 per cent perfect, and that an average tar may contain possibly 2.0 per cent of light oil, but this figure represents an amount so small that if we had to depend upon the toluol in coal tar for our supply of T.N.T., we would be in bad straits.

Analyses of tars as performed in the laboratories show that the tars begin to distill at about 170°C., which is about the point where light oil leaves off. A fairly representative picture of the chemical composition of tar may be obtained from Table 1, prepared by Weiss and Downs. Their work was reported in 1923 before the American Chemical Society. It should be borne in mind that the figures apply to amounts of substances actually present, and not to amounts recoverable commercially by the average tar plant.

TABLE 1.—*Constituents of Tar*[a]

	Dry Tar, Per Cent by Weight
Light oil:	
Crude benzene and toluene	0.3
Coumarone, indene, etc	0.6
Xylenes, cumenes, and isomers	1.1
Middle and heavy oils:	
Naphthalene	10.9
Unidentified oils in range of naphthalene and methylnaphthalene	1.7
Alpha monomethylnaphthalene	1.0
Beta monomethylnaphthalene	1.5
Dimethylnaphthalene	3.4
Acenaphthene	1.4
Unidentified oil in range of acenaphthene	1.0
Fluorene	1.6
Unidentified oil in range of fluorene	1.2
Anthracene oil:	
Phenanthrene	4.0
Anthracene	1.1
Carbazole and kindred nonbasic nitrogen containing bodies	2.3
Unidentified oils, anthracene range	5.4
Phenol	0.7
Phenol homologues (largely cresols and xylenols)	1.5
Tar bases (mostly pyridine, picolines, lutidines, quinolines and acridine)	2.3
Yellow solids of pitch oils	0.6
Pitch greases	6.4
Resinous bodies	5.3
Pitch (460°F. melting point)	44.7
Total	100.0

[a] Weiss and Downs: Amer. Chem. Soc. (1923).

The chemicals of major commercial importance in coal tar are naphthalene, phenol, the cresols and the xylenols. For these there is an active market. Of lesser importance are the tar bases (the higher boiling pyridines, quinoline and its homologues), anthracene, acenaphthene, carbazole, fluorene, phenanthrene and the methylnaphthalenes.

The giant among coal-tar chemicals is naphthalene. This material, which is present in tar to the extent of around 10 per cent is recoverable commercially to the extent of about 5 per cent.

Its present value is in the neighborhood of 2.75¢ per pound in crude form, which is only 2° from the melting point of the pure product. Statistics show that approximately 166 million pounds were used in this country in 1940; only 6.3 million pounds of this total were imported. About half of the naphthalene used in that year was charged in crude form into the vaporizing chambers of the phthalic anhydride plants of many chemical companies. In the neighborhood of three fourths of the phthalic anhydride consumed in 1940 was utilized in the manufacture of alkyd resins and at least one tenth in the production of phthalate plasticizers. The bulk of the remainder was used in the production of phenolphthalein, benzoic acid, and anthraquinone.

TABLE 2.—*Consumption of* 166 *Million Pounds of Naphthalene in* 1940

	Calculated as Crude, Per Cent
Phthalic anhydride	50
Other intermediates	15
Moth proofing	12
Miscellaneous	23
	100

The most important recent development in the naphthalene field has been the enormous increase in demand in the chemical industry. It is probable that its production for 1936 in the United States was around 75 million pounds and it is estimated that the 1942 figure will reach 200 million pounds. Previous to the beginning of the present World War, naphthalene was sold in this country from England and Germany at such a low price that it did not pay the tar distillers in America to recover the naphthalene content. Since that time, however, the demand for naphthalene has been great, and with the shutting off of the imports from abroad the output of naphthalene has been increased very largely.

Phenol and Other Tar Acids in Defense

Phenol promises to become one of the most important chemical raw materials in the whole problem of defense, and in the chemical field. Until recently (except in 1917–1918) the major portion used was secured from coal tar, but recent demands and war uses have increased the synthetic production of this material from benzene, so that today virtually three fourths of that used is so secured, and large new plants are building or are projected.

During the last war there was a very urgent demand for phenol for manufacturing picric acid, one of the commonly used explosives of that war, and this use constituted one of the major applications for tar acids at that time. Now, in 1941, tar acids have taken a much firmer place in defense. While picric acid has been largely displaced by T.N.T. it still enjoys a number of specialty uses—for example, as the salt, ammonium picrate, for the bursting charge in armor-piercing shells—but tar acids have proved so indispensable since 1917 that now, in various forms, they go under water with the submarines, overhead with the airplanes and ride with the troops in their jeeps and tanks, as phenolic resins produced from phenol and its homologues and for the main part combined with formaldehyde.

Shock resistance of a construction material is considered of primary importance by the Navy in consideration of the great stresses suddenly applied by the detonation of heavy guns and high explosive charges in the vicinity of combat equipment. Instrument cases, terminal blocks for an infinite variety of important electrical connections and junction boxes for submarines are types of equipment that must be strong. The Bureau of Ships, Navy Department, has written specifications for such items of equipment to be made from phenolic resins. Similarly, parts used by the Army include instrument cases, fuse timers for shells and electrical fittings for tanks and trucks.

Rapid construction of temporary buildings and cantonments is always a prime requisite in countries preparing for war. This need is now met by plywood panels, which simplify construction and provide strong weatherproof and verminproof structures by virtue of the phenolic-resin plastic bond between the sheets of plywood.

In the air, where every pound and every ounce of weight have tremendous influence on performance, plastics made from phenols are essential materials. The pilot in the air talking with other ships and with his ground radio station uses antenna masts insulated by means of a base of phenolic resin. This material is employed in the radio housing and the radio sets inside the plane, as well as the pilots' switch panel, which carries the controls for landing flaps and wheels and for the electrical circuits. Engineers in research discovered long ago—as time is measured in air-transportation circles—that plastics are corrosion-resistant. The control cables that extend from cockpit to tail surfaces and laterally out toward the tips of the wings operate on corrosion-resistant pulleys and bushings made of laminated phenolic plastics. A new tar-acid product, dibutyl-meta-cresol, keeps the airplane hose lines of synthetic rubber soft and plastic and thus plays a role similar to that of tricresyl phosphate, also made from cresol, which is given the job of keeping plastic the waterproof resin coatings used on the electrical cables on warships and submarines. Airplane fuselages and wings of plywood bonded with phenolic resin may soon be common.

In our enthusiasm for the finished airplane, the tank and the shell, we must not forget that behind them lie the factories that produced them. In these factories, as in the finished product, tar acids play an important part. In the nation's shipyards, batteries of high-speed swing grinders mounting 24-in. wheels bonded with

phenolic resin are shaping the contours of massive propeller castings, gun turrets and large deck fittings. These grinding wheels have two basic ingredients—the abrasive grains and the phenolic resin that holds these grains firmly in place while they do their work. So fast do these grinding wheels turn that approximately 45,000,000 cutting points engage the work every minute. Steel bodies for high-explosive shells are being machined today more than 20 times as fast as during the World War. This has been achieved, at least in part, by the wide use of high-speed, tungsten-carbide-tipped tools in the various tooling setups. Like any other cutting tool, tungsten carbide tools must be sharpened, and this is being done by grinding wheels made of minute diamond particles bonded with phenolic resins.

The applications mentioned above are typical of the diversified ways in which tar acids play their part in defense. So vital is their part that after Dec. 1, 1941, no deliveries of phenol, one of the tar acids, could be made except as specified by the Director of Priorities.

One cannot leave the subject of phenol without calling attention to the developments made by the du Pont company in the manufacture of Nylon. According to a definition and information given by Dr. J. S. Beekley:

"Nylon is the generic name given by the du Pont company to the synthetic linear superpolymers developed by its research on the building of large molecules, or super-polymers, from small molecules. Nylon is officially and scientifically defined as 'a man-made, protein-like product (polyamide) which may be formed into fibers, bristles, sheets and other forms which are characterized when drawn, by extreme toughness, elasticity, and strength.'

"The term Nylon does not refer to the material in any one form, nor does it refer to any one chemical compound, but rather to a family of related compounds, of which there are many members.

"The particular type of Nylon that at present appears most suitable for making textile yarn is made from a diamine and a dibasic acid, both of which are derived from phenol, which in turn is derived from coal, either as a by-product in coking operation or from benzene, a by-product in coking coal. Oxygen from the air is used in making the acid and ammonia, a synthetic product obtained as we have seen from coal, air, and water, is used in making the diamine. Various reactants and solvents used in the intermediate process steps have their origin in coal. So this particular Nylon is obtained from coal, air, and water, and the chemical operations involved in its making are carried out at the Belle, W. Va., works. Final polymerization and spinning are carried out at the company's plant at Seaford, Delaware."

Thus we see that one other great new product, while commonly spoken of as being made from coal, water, and air, is really a product of phenol from coal carbonization.

Road Tar

The preparation of road tar involves the transportation and handling by the tar distiller of a large quantity of tar and its selective blending to make fluxes or finished products, and also the technical supervision of quality to ensure a minimum variation in the properties of the products. Finished road tars are shipped in tank cars, tank wagons and drums. Coal tar makes an excellent road and is cheaper than concrete. The building of hard-surfaced roads is a major industry and the fact that 200,000,000 gal. of road tar is now being used annually makes an important outlet for the heavy ends of coal-tar distillation, thereby acting as a complement to the production of creosote oil, which is made up of the lower boiling fractions. The greatest quality advantage of coal tar over competitive materials is as a skid resistant. Roads built with tar are much more desirable from this standpoint than the competitive materials, cement and asphalt. Great improvement has been effected in the methods of building and application of tar in building and resurfacing, and the days of tar damage to cars and tires during

road construction and repair have largely passed. More than 40,000 miles of skid-resistant roads are built and maintained each year with such materials, and it is used also for surfacing of air fields and auto-parking lots.

Creosote Oil

Not often considered, but of vital importance to the transportation and communication industries are the creosote oils used in wood preservation. Last year approximately 175,000,000 gal. of creosote was used in treating railroad ties and other wood products. The railroads learned long ago that much money could be saved and additional safety provided by using creosoted ties. U. S. Government reports say that millions of acres of timber per year are saved by the use of pressure-treated creosoted ties on railroads. Replacements of ties in the United States are virtually all creosoted. General industry has waked up to the value of wood preservation and its use is extended to telephone and telegraph poles, fences and fence posts, wharves, deck bridges and culverts, mine timbers and ties. As wood becomes more expensive, and labor also, these costs are offset by the longer life of the material and increased safety in use.

Insecticides

An unceasing, serious warfare must be waged by mankind against the insect world. The Mediterranean fruit fly, the boll weevil, the corn borer, the Japanese beetle, the potato bug, aphis, grasshopper and chinch bug are but a few of the pests that have made great inroads on agriculture during the past few years. The by-product coke industry is developing effective munitions from coal products with which to fight these devastating hordes, largely from light oil and tar derivatives.

More recently, the discovery and application of the manufacture of thylox sulphur from the hydrogen sulphide, in the gas coming from the ovens, has opened up a new field in the successful development of fruit, in destroying scale and other fungi. This sulphur is so finely divided that it will remain in suspension in water and may be sprayed on the fruit trees, preventing scale and fungus and allowing the fruit to develop to perfection. This field for sulphur promises to be a most important help to the farmer and fruit grower. The demand far exceeds the present supply.

Sanitation, Disinfectants and Medicines

The importance of coal products in determining the cause of illness; in prevention, diagnosis, treatment, and cure of disease and in the alleviation of suffering, can scarcely be overestimated. Since Lord Lister in 1867 introduced phenol to prevent blood poisoning, countless lives have been saved by that product alone. Time does not permit a review of the romantic and colorful role played by the aniline dyes, obtained from benzol in the history of medical bacteriology. The widespread use of coal-tar crudes and intermediates such as phenol and cresylates in household, industrial and agricultural sanitation are too well known to require discussion.

Beginning with the use of coal-tar dyes in staining microscopic slides, it has been possible to develop the whole matter of the study of disease germs and of effecting cures for such afflictions.

One of the most dramatic chapters in the story of chemicals made from the products of coal carbonization is that dealing with medicinals. Many of the 145 every-day medicinals whose origin can be traced back to coal are old friends whose names we recognize at once as aspirin, acetanilide, antipyrine, novocaine, saccharine, salicylic acid and hexylresorcinol; others like sulfanilamide, sulfapyridine and sulfathiazole are new and the cures they have effected in staphylococcal, streptococcal and pneu-

mococcal infections, such as pneumonia, meningitis, erysipelas and septicemias have been so sensational that time and again they have been mentioned in the daily press. The story of medicinals would not be complete without mention of the vitamins. Some of these are already being synthesized from coal-tar and light-oil chemicals and as time progresses they will become available in larger quantities and, it is hoped, at lower prices, for the benefit of mankind.

Pitches

Coal-tar pitches serve many important engineering, building and manufacturing industries. Brick pavements are filled with pitch compounds; roofs, foundations, walls, bridge decks, and abutments are protected against water with coal-tar pitch; pitch is used as a binder for coal briquettes, carbon electrodes, battery boxes, foundry cores and even for the clay pigeons that are used as targets for shotgun practice; pitch is used also as fuel in pulverized or liquid form, especially when freedom from ash and low sulphur content are important, as in the burning of lime or the making of sodium silicate. In some of these applications, coal-tar pitch is peculiarly an effective binder for such aggregates or fillers as coal, coke, cotton linters, sand and clay. In other applications, such as roofing, waterproofing, and the impregnation of fiber conduits, it is used because of its superior water-resisting characteristics. No bituminous material with which we are familiar is better able than coal-tar pitch to resist the chemical and physical alterations that may be caused by water. For this reason, coal-tar pitch is meeting with increasing favor for the protection of flat roofs that are subjected to contact with water or snow for long periods of time. In fact, the producers of coal-tar pitch feel so confident about the water resistance of their product that they guarantee for 10 to 20 years flat roofs that are to serve as spray ponds or as reservoirs for water used in air-conditioning systems.

One of the latest developments in building construction is the use of a 2-in. water layer on the roof to help cool the interior of the building. It is even expected that the flat-roofed houses of the future will be cooled in this manner in the summer and that the same flooded roofs will become skating rinks in the winter. This possibility is receiving serious consideration.

Pitch Coke

As these uses for pitch multiply, they will provide outlets for greater proportions of the residues from the distillation of tar. The lack of suitable outlets for pitch has always been one of the problems of the tar-distilling industry. In fact, only a few years ago frequent references were made to the burden of pitch. To some extent at least that burden has been relieved by the development of methods for making coke from coal-tar pitch. The Koppers method for accomplishing this consists in spraying pitch of high melting point, while in a molten condition, into a heated by-product coke oven. The pitch is heated and partly distilled during the charging period and is further distilled and transformed into coke during a subsequent heating period. When completely coked, the charge is pushed from the oven in the ordinary manner and with the regular pushing and handling equipment. Coke produced in this manner, because of its high degree of purity, high density and low volatile content, is especially suitable for metallurgical electrodes and as a carburizer in open-hearth furnaces using large percentages of scrap.

Paints, Varnishes and Lacquers

In no branch of industry has the synthetic production of new products been more effective in changing the entire viewpoint than in the manufacture of varnishes, lacquers, and similar materials. A very

large proportion of the alkyd resins produced from phthalic anhydride, originating in naphthalene, is being used in this field, the results of which are most satisfactory. While the nitrocellulose lacquers have not lost their hold on the builders of high priced automobiles, the alkyd resin groups are doing the bulk of the business. Paints and varnishes containing plastics are in great demand because of their quick drying, vivid coloring and absence of odor.

Rubber Industry

Crude rubber, as imported, is of little value except for making rubber cements and adhesives, because it softens and loses its shape if warmed. When heated with sulphur, however, marked changes occur and it becomes tough, elastic, and very strong. When further treated, before vulcanization, by incorporating into it various minerals, organic chemicals, and carbon black, rubber becomes well fitted for the many uses made of it.

The part played by the organic chemicals is noteworthy and is reflected in the recent improvements in the quality of automobile tires and in the speed with which they are manufactured. Not many years ago it was necessary to heat tires for a long time in order to vulcanize them. This prolonged heating injured the rubber, so that the tires were very short-lived. The escape from this dilemma came with the discovery of vulcanization accelerators. These are organic chemicals and the majority are made from benzene, toluene, xylene, or naphthalene. Through their use, tires once requiring 10 per cent of sulphur and 6 hr. or more of heating are now vulcanized with 3 per cent of sulphur in 45 min. Not only has this brought about an improvement in the quality of the tires but tire manufacturers are saved an estimated additional capital investment of 50 million dollars, since each mold now produces five to ten times more tires than formerly.

These improved tires were still unsatisfactory in that when exposed to sunlight they hardened and cracked. This behavior was traced to the action of the air on the warm rubber and led in time to the development of other chemicals called antioxidants. Most of them are made from benzene, toluene, or naphthalene. They possess the property of retarding oxidation by air and of providing great resistance to deterioration arising from heat and flexing. If it were not for the use of these antioxidants in the modern tire, the thin side walls of low-pressure tires, which are in an almost constant state of flexure, would crack through much sooner, and the inner tubes of bus and truck tires would melt or become ruined by the intense heat developed within them. The antioxidant chemicals in a ten-dollar tire may not cost more than ten cents, but they increase the life of the tires at least twofold.

Owing largely to the use of vulcanization accelerators and antioxidants, the life of the automobile tire has increased to the point where today performance records of 15,000 to 20,000 miles are common. This compares with the 2,000 to 5,000 miles formerly obtained.

In addition to rubber, sulphur and organic chemicals, finished rubber products may contain comparatively large quantities of various materials referred to as fillers, softeners, extenders, and tack producers. Carbon black and zinc oxide are widely used fillers. Coal-tar pitch and asphalt are used as extenders and also to give the rubber product certain desired properties. Stearic acid, pine tar, coal-tar distillates, and to a smaller extent coal tar itself, are used as softeners. The coal-tar distillates are comparatively high-boiling distillates and are used because of their solubility in rubber and their ability to wet pigments. These properties greatly promote the uniform dispersion of pigments in rubber and speed up the milling and mixing operations. Coumarone-indene resin made from coal-tar solvent naphtha

is another common softener. In addition, it functions as a tack producer and as an extender. Other products of coal-tar origin enter rubber products in the form of dyes and colors and as complex sulphur-containing chemicals, which sometimes are used instead of sulphur for vulcanizing high-priced white stocks. Toluol and xylol are used as rubber solvents for making dipped goods and in the manufacture of rubber adhesives and cements. Cresol, cresylic acid and coal tar are used in reclaiming rubber. Benzol promises to become a major necessity in the manufacture of synthetic rubber. Although the butadiene can be produced from oil gases, the styrene, with which the butadiene is polymerized later, must be made from benzene.

Explosives

Strictly speaking, the matter of the by-product coke oven and its place in defense are usually considered from the standpoint of toluene and T.N.T., its explosive derivative. The great explosive in modern war is T.N.T. (Tri Nitro Toluol). The by-product industry produces approximately ½ gal. of toluol per ton of coal carbonized, which when nitrated is a large source of T.N.T. Plants are being constructed to produce toluol in large quantities from petroleum by new processes but a large portion of that needed will be derived from distillation of coal. T.N.T. is the best explosive because, though it is most powerful, it is fairly safe to handle. Shells of T.N.T. can be loaded by melting the material, which can be done at a lower temperature than that of boiling water. It must be detonated by a special cap. It is not corrosive.

During the previous war, much picric acid made from benzol was used; but little, comparatively, is used at present because it was found that picric acid corrodes the shells, making dangerous and unstable substances, though Russia is still using picrate and a way has been found to prevent corrosion by means of a plastic coating inside the shell.

Merchant Coke and Gas Plants

While the iron and steel industries are the largest owners and operators of coke-oven plants, the gas-manufacturing and merchant plants are important producers of coke, operating nearly one fourth of the total capacity. Many such plants produce coke-oven gas as the major product, heating the ovens with producer gas made from coke, thus giving a large gasmaking capacity. The coke produced is an ideal solid fuel suitable for domestic use. In the East, plants of this type have been built at Boston, New Haven, New York, Brooklyn, Jersey City, Camden, Philadelphia and Swedeland.

The production and distribution of domestic coke therefore has become a major undertaking and supplies to householders of these cities a clean, efficient, low-ash and satisfactory fuel amounting to nearly 3,000,000 tons per annum. These plants also make foundry and other industrial cokes and a large amount of coke for water gas and producer-gas operations.

Coke Ovens in the Prevention of Air Pollution

Inhabitants of the section east of the Mississippi where soft coal is burned are becoming air-pollution conscious, and since the cleaning up in St. Louis has been so well advertised in the press a great many other cities and towns are giving the matter much attention. The problem is a real one and must be faced, but it is not an easy one to solve. At present, nearly 80 per cent of the homes in Pittsburgh are heated by the use of high-volatile bituminous coal. On a cold, damp, still morning in Pittsburgh, more than one hundred thousand domestic chimneys send forth various amounts of smoke, which lies in the river valleys and makes impossible conditions

for decent living. One solution is in the underfired stoker or some new type of smokeless stove, but this involves an expenditure on the part of the householder, which is not always possible.

There is little question of the helpfulness and economy of the underfeed coal stoker. It would seem that some fuel must be found that can be supplied in quantity and at not too high a price. At the present time (1941 Pittsburgh) a good grade of coal can be put in the cellar at about 60 per cent of the cost of by-product coke. On the face of it, this looks as though the householder, if compelled by law to burn this smokeless fuel, would do so at considerable expense. This viewpoint, however, is exaggerated by the price differential, for the average domestic furnace or stove secures an efficiency of between 45 and 50 per cent whereas coke can be burned to secure 70 per cent efficiency. This being true, the home that burns 10 tons of bituminous coal would burn only 7 to 8 tons of coke, so that the increase would amount to approximately 15 per cent during the season. This would seem to be a small amount when considered in connection with improvement in health and contingent advantages to the community in reducing cleaning, painting and general costs of upkeep.

There has been considerable controversy, both popular and technical, about the processing of high-volatile coal at low temperatures to manufacture a product that can be burned without producing smoke. Millions of dollars have been lost in trying to develop such processes. The nearest thing to a smokeless fuel in this country has been worked on for many years by one of the largest coal companies in America, with a certain amount of success. The fuel is known as Disco. It is not a cheap fuel and has advantages and disadvantages. It is easily kindled and holds a fire remarkably well. Its chief disadvantage is that, owing to its high combustibility, it gives off gases, which escape unburned and lower the efficiency of its combustion. For an open grate, it is ideal.

In the author's opinion, by-product coke, made either in ovens of the present type or by the use of a comparatively new type, is the answer to the domestic fuel problem, after those who can afford stokers and all who can afford gas have been satisfactorily sold. To make this, there must be a profitable disposal of the resulting gas.

This whole matter is worth the deepest consideration—the pollution of our atmosphere must be stopped.

It has been customary to blame the "smog" on industrial plants. While undoubtedly closer, honest inspection and rigid rules would bring considerable result, the real answer lies in smokeless domestic heating. This is not propaganda for the by-product oven. The solution must be on a plane above prejudice and self-interest, and should receive the best thought and action of the community.

Fertilizer Industry

Finally, there must be mentioned one more great industry which pays homage to the coke oven; namely, the fertilizer industry. Nitrogen is one of the elements essential to plant growth. Sulphate of ammonia, formed from ammonia and sulphuric acid in the by-product coke plant, is the largest contributor of this essential material. In 1941, close to 900,000 tons of sulphate was produced, which will in turn be mixed with phosphate and potash and be placed on thousands of acres of land to increase and maintain the crop.

Ammonia is used also in the production of nitric acid, one of the raw materials in the manufacture of T.N.T. Little ammonia, however, is directly received from the coke-oven plants for this purpose; it is produced as synthetic ammonia from blue gas and producer gas made from by-product coke. A plant for this purpose is being built at Morgantown, W. Va. Similar

plants are in operation at Belle, W. Va. and Hopewell, Virginia.

Summary

In summary, the by-product coke plant is an essential factor in the following industries and occupations, upon which innumerable other industries, and our war effort especially, depend:

1. Coal mining.
2. Iron and steel.
3. Gas manufacturing.
4. Road building.
5. Wood preserving.
6. Railroad building.
7. Dyestuff.
8. Plastic.
9. Chemical.
10. Paint, varnish and lacquer.
11. Explosive.
12. Fertilizer.
13. Photographic chemical.
14. Leather.
15. Perfumery.
16. Textile.
17. Medicinal and sanitation.
18. Drug.
19. Tire manufacturing and rubber.
20. Petroleum refining.

Acknowledgments

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Research on Coal for Domestic Stokers

By Walter Knox,* and J. D. Doherty,† Member A.I.M.E.

(Easton Meeting, October 1941)

In 1939, at the request of The Koppers Coal Co., the Koppers Company Research Department established a Stoker Coal Research Laboratory for the purpose of investigating the performance characteristics of stoker coals in typical domestic stoker installations. Fig. 1 is a photograph of part of the laboratory.

The material in this paper deals with the performance of one coal in a single installation. Presented are details of this installation, mechanical characteristics of the stoker, and the results of a series of burning tests in which adjustments were systematically varied.

Summary of Results

Mechanical Features

1. A large percentage of the total power consumed by a domestic stoker is used to operate the gear case and motor (Fig. 3).

2. The power consumption per ton goes down with an increase in rate of coal feed (Fig. 4).

3. The automatic air control in this stoker showed advantages over manual control (Fig. 5).

4. The air velocities through the retort tuyeres were uniform around the retort, but varied between the rows of tuyere rings (Fig. 6).

5. The crushing of coal with this stoker increased the minus ⅛-in. from 11.6 per cent to 19.2 per cent. There was a noticeable segregation of coal as it came from the retort and distribution was not uniform (Fig. 7).

6. Measurement of air entering over the fire showed that with some not uncommon adjustments this air may be a high percentage of the total air supply (Fig. 8 and Table 1).

Operating Performance

1. The removal of clinker was shown to give a period of poor performance characterized by a high fuel-bed resistance and a high percentage of carbon monoxide in the flue gases (Fig. 10).

2. The fuel-bed resistance was found to decrease as the fuel bed built up in a normal response to heat demand after a prolonged hold-fire (Fig. 11).

3. With intermittent operation the burning rate was almost always less than the feeding rate. The wide variations were found to be greatly dependent upon the length of the On period and the length of the intervals between On periods (Fig. 12).

4. The maintenance of a low furnace draft led to less fly ash, better clinker formation, and higher efficiencies (Fig. 13).

5. The maintenance of a minimum of excess air also resulted in less fly ash, better clinker formation, and higher efficiencies (Fig. 14).

Guiding Principles

In planning the investigations, the following principles were adopted:

Selection of Heating Equipment.—Use only standard models of stokers and furnaces representative of those widely used.

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Size of Equipment.—Select small units such as would be found in one-family houses, since experience has shown that the smallest stokers are likely to be the most sensitive to changes in fuel. Larger installations usually can take a wider range of coals and also frequently have more skilled attention.

Installations.—Install equipment as prescribed by manufacturers.

Draft.—Use an induced-draft fan to avoid the uncontrolled variable of chimney draft. Use conventional automatic draft controls in regulating to duplicate as far as possible draft characteristics of typical house-heating plants.

Instruments.—Apply testing devices to give the maximum amount of information without interfering with normal operation of the burning equipment. Use recording instruments to obtain continuous records for comparison with a photographic record of the fuel bed.

Test Procedure.—Use a timer to obtain various rates of operation. Include extreme conditions—a long steady-running period and a low-rate hold-fire. Make tests long enough for normal clinker formation and measurable accumulation of fly ash.

Coal.—Adopt one coal as a standard and use it for tests of variables in burning equipment, such as stoker adjustments.

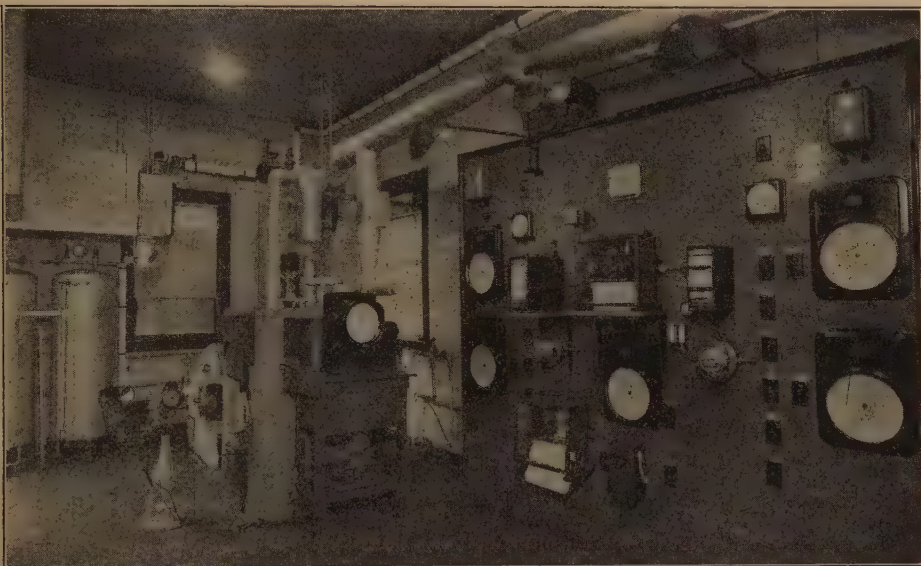

FIG. 1.—PART OF THE RESEARCH LABORATORY

Automatically Operated Equipment and Instruments.—In order to require as few men as possible and to make shift work unnecessary, plan tests and provide sufficient automatic equipment to record desired information (including appearance of fuel bed) with no one in attendance overnight or during week ends.

BURNING EQUIPMENT

The burning equipment in the laboratory consists of standard models of small domestic stokers installed in both steam and warm-air house-heating furnaces. Discussion in this paper will be confined to a single installation using the standard coal.

The stoker in this installation was a hopper model with a continuous-feed gear case. It had three coal-feed rates, ranging from 15 to 30 lb. per hour. The air from the

fan was controlled by an automatic air regulator. The stoker was installed in a sectional cast-iron boiler rated at 600 sq. ft. of steam radiation. Fig. 2 shows details of this installation.

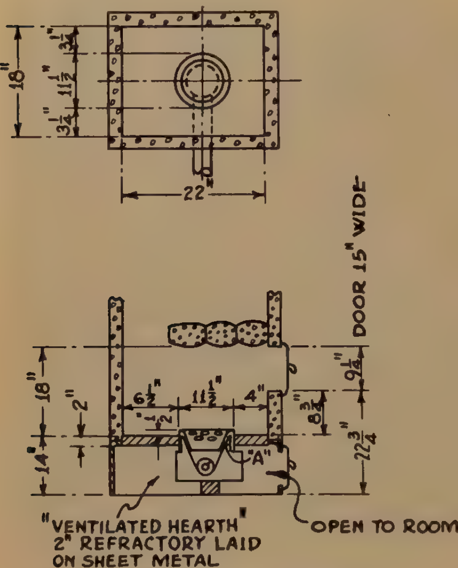

Fig. 2.—Installation of stoker No. 1 in steam boiler.

Boiler rated at 600 sq. ft. steam (outlet). Boiler heating surface, 35.7 sq. ft.; hearth area, 2.75 sq. ft. "Head room" as installed-18 in.; combustion space, 4.125 cu. ft.

A shows position of thermocouple embedded in the inside casting 2¾ in. below top of retort.

The boiler was chosen to represent the typical hand-fired heater converted to stoker firing found in numerous small homes. Better results could have been obtained with a larger boiler or one specially designed for stoker firing.

Preliminary Tests

Although the chief purpose of these investigations is to study the performance of coals in stokers rather than to study stoker design, it was considered important to determine the mechanical and electrical characteristics of the stokers used, since resnlts are affected by design factors. Therefore, before the stokers were installed, tests were made to determine the power requirements, the characteristics of both automatic and manual air controls, the air distribution through the tuyeres, the crushing of coal by the feed screw, and the distribution of coal from the retort.

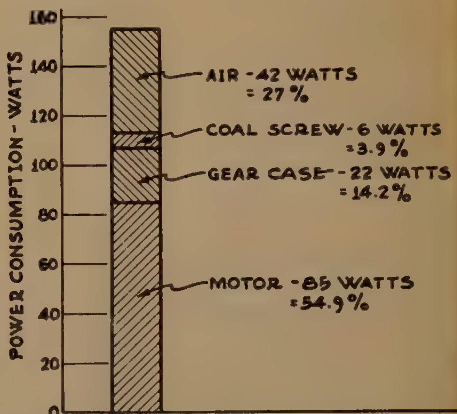

Fig. 3.—Breakdown of power consumption

Feeding 33.7 lb. coal per hour and 90 cu. ft. per minute of air.

Power Consumption.—Fig. 3 shows a breakdown of the power consumption when the stoker was feeding 33.7 lb. of coal per hour and delivering 90 cu. ft. of air per minute against the retort resistance only. In making this chart it was assumed that motor and gear-case losses did not change appreciably under the loads imposed. With a large percentage of the total power required to operate the motor and gear case at no load, it is apparent that decreasing the coal-feed rate (and thus operating the stoker a greater part of the time) will increase the power consumption per ton of coal. The relation of total power consumption to coal-feed rate is shown in Fig. 4. In determining this curve the volume of air delivered by the fan per pound of coal fed was kept constant.

Comparison of Automatic and Manual Air Controls.—Fig. 5 shows the variations in air delivery when the resistance was increased by means of a metal plate over the retort. With this stoker operating on the automatic air control, there was com-

paratively little variation in the air delivered over rather a wide range of resistance, while with the manually-set fixed damper in the air duct the amount of air delivered decreased regularly with increase of resistance, as would be expected.

Two lines are shown for each of the automatic air-control settings. As the arrows indicate, one was obtained when increasing the resistance and the other when decreasing it. At lower air deliveries (not shown) the volume of air was more nearly constant and the difference between the curves for increasing and decreasing the resistance was much less.

Air Velocity through Tuyeres.—The retort of the stoker had a total of 9.25 sq. in. of free tuyere-port area. There were two rows of inside ports and one row of outside ports. The air velocities measured at the various tuyere ports are shown on Fig. 6 for one rate of air delivery and with no coal in the retort. Air velocities were very uniform around the retort. The air delivery was distributed as follows: outside ports, 15 per cent; upper inside ports, 30 per cent; lower inside ports, 55 per cent.

Coal Crushing and Distribution.—In Fig. 7 are presented data obtained on the crushing and distribution of coal. The rotation of the feed screw appears to have an important bearing on the coal distribution, more coal being delivered to the left because of the counterclockwise rotation of the feed screw. The amount of crushing and the delivery of a higher percentage of fines to the quadrants farthest from the hopper agree with published results of other tests on domestic stokers.[1]

Over-fire Air.—The quantity of air entering the furnace over the fire with

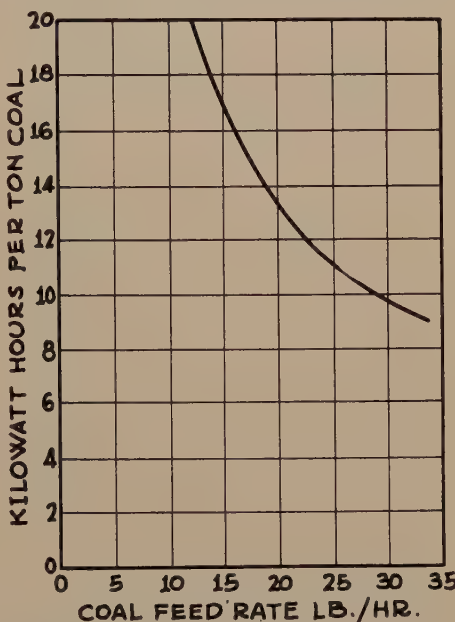

FIG. 4.—RELATION OF POWER CONSUMPTION TO COAL-FEED RATE.
Air-coal ratio constant.

various drafts was measured by means of a test-orifice set-up. A connection was made to the smoke outlet of the boiler and air was drawn through the orifice by means of a fan. The retort and tuyeres were sealed to prevent air from being drawn through the stoker. The results are shown graphically in

TABLE 1.—*Air Supply to Fire with Different Furnace Drafts, Stoker Running*

Draft, in. of water	0.02		0.07		0.12	
	Cu. Ft. per Min.	Per Cent	Cu. Ft. per Min.	Per Cent	Cu. Ft. per Min.	Per Cent
Air:						
Stoker fan	41	84	42	72	43	66
Over fire	8	16	16	28	22	34
Total	49	100	58	100	65	100

[1] References are at the end of the paper.

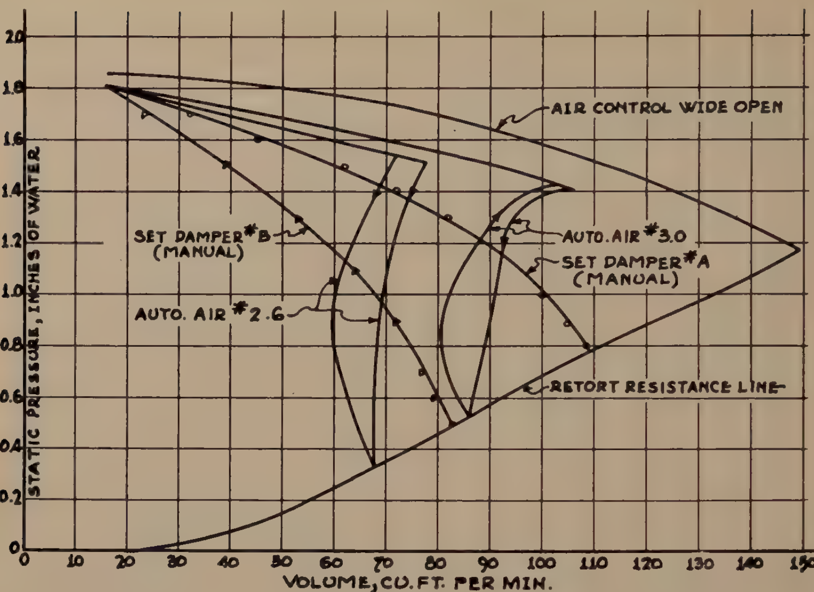

FIG. 5.—COMPARISON OF AUTOMATIC AND MANUAL AIR CONTROLS.

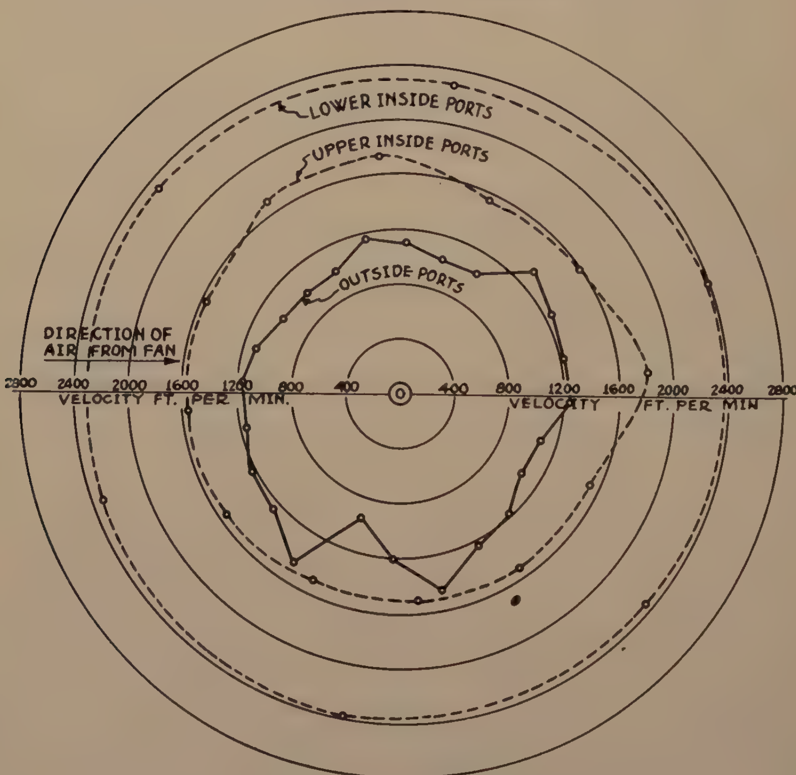

FIG. 6.—AIR VELOCITY THROUGH TUYERES.
Total volume of air 93 cu. ft. per minute.

Fig. 8 while Table 1 shows a comparison of the volumes of air delivered by the stoker fan and the quantities drawn in over the fire for three tests at different furnace drafts.

Coal Used

A commercially prominent double-screened low-volatile stoker coal produced from the Pocahontas No. 3 seam, in McDowell County, West Virginia, has been adopted as the standard, and was used for all tests reported in this paper. It is mechanically cleaned and oil-treated.

So that the test coal would be similar to that delivered to consumers, a car of coal was shipped to the laboratory. After a normal amount of handling, several tons were bagged and stored for test work. Data on the coal used in the stoker tests are presented in Table 2.

Table 2.—*Coal Used in Tests*

	Per Cent
Moisture	1.0
Proximate analysis (dry basis):	
Volatile matter	16.4
Fixed carbon	77.8
Ash	5.8
Ultimate analysis (dry basis):	
Carbon	86.4
Hydrogen	4.3
Nitrogen	1.1
Oxygen	1.8
Sulphur	0.6
Ash	5.8
B.t.u. per pound, dry basis	14,860
Ash-softening temperature, deg. F.	2330
Dustiness index	34
Nominal size, in.	¼ to ⅝

	Per Cent
Screen Test:	
+⅝-in. round	0.2
⅜ to ⅝-in. round	30.5
No. 4 to ⅜-in. round	38.3
No. 8 to No. 4	19.4
No. 28 to No. 8	7.1
Minus No. 28	4.5
	100.0

Test Procedure

A standard test schedule was adopted, which requires 5½ days and includes continuous, intermittent and hold-fire operations (Table 3).

During the 12-hr. continuous run and the two 5-hr. intermittent runs of this standard schedule, complete data, including analyses of flue gas by an Orsat apparatus, are taken. During overnight and other periods only the recording instruments are used, except that a record is made of the water evaporated, the revolutions of the feed screw, and

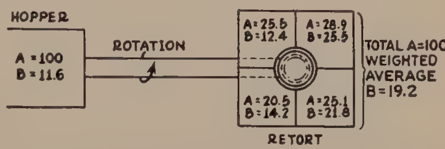

Fig. 7.—Crushing of coal and distribution from retort.

A, percentage of total coal; *B*, percentage minus ⅛-in. of portion.

the watt-hours consumed. Each test is started with a clean hearth and clean boiler passes. Clinkers and fly ash are allowed to accumulate during the entire

Table 3.—*Standard Test Schedule*

	Elapsed Time		Running Time of Stoker	
	Hr.	Min.	Hr.	Min.
Tuesday afternoon:				
Start fire with clean hearth	0	30	0	30
Overnight: 15 min. on, 45 min. off each hour	14	00	3	30
Wednesday morning:				
Preparatory run—continuous	1	00	1	00
Continuous efficiency test	12	00	12	00
Overnight: 10 min. on, 50 min. off each hour	13	30	2	14
Thursday morning:				
Response—continuous	2	00	2	00
Intermittent: 20 min. on, 40 min. off each hour	5	00	1	40
Overnight hold-fire: 5 min. on, 55 min. off each hour	17	00	1	26
Friday morning:				
Response—continuous	2	00	2	00
Intermittent: 20 min. on, 30 min. off each h ur	5	00	2	30
Friday afternoon to Monday morning:				
Hold-fire: 3 min. on, 57 min. off each hour	65	00	3	20
Monday morning:				
Response—continuous	1	30	1	30
Total	138	30	33	40

Running time, per cent of elapsed time........ 24.3

5½ days and during this time the fire is not touched. This does not cause an excessive accumulation of ash and clinker because the hearth is clean at the start of the test and the ash content of the coal is

not high. In normal operation, after building up the usual ash bed, clinker would be removed oftener. At the end of the test, clinker, coke and loose ash are removed from the firebox and weighed. Fly ash also is cleaned out and weighed. All tests reported herein were run with a coal feed of 15.5 lb. per hour.

Data Taken

The data taken for each test includes:

Coal.—Moisture, proximate analysis, ultimate analysis, ash-fusion temperature, B.t.u., cubic foot weight, screen test, dust test.

Test Data.—Barometer, relative humidity, room temperature, water temperature, coal fed, revolutions of feed worm, running time, power demand* total power, air delivered by fan, water evaporated, boiler output,* furnace draft,* stack draft, windbox pressure,* flue-gas temperature,* retort temperature,* flue-gas analysis Orsat (CO_2, CO, O_2) CO_2 recorder,* smoke,* photographic records.*

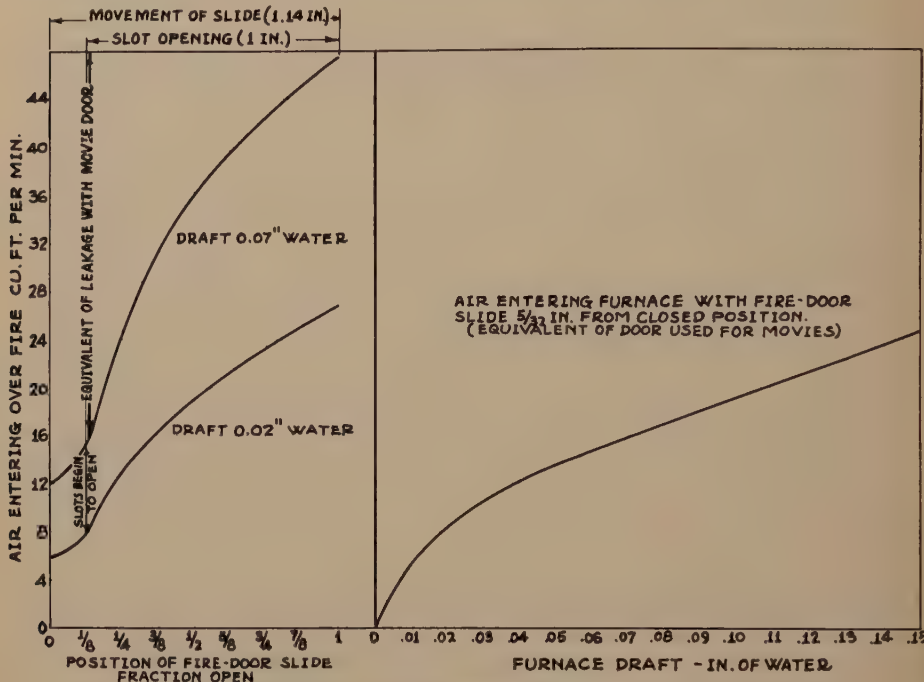

Fig. 8.—Air entering over fire.

Material at End.—Clinker, weight and percentage of ash; loose ash, weight and percentage of ash; coke, weight, proximate analysis, gravity; fly ash, weight, cubic foot weight, screen test, percentage of volatile matter, percentage of ash. Table 4 is a typical condensed data sheet, giving totals and averages for each period and for a complete test.

Standard Methods of Obtaining Data

Coal Fed.—The quantity of coal fed is determined by leveling the coal in the hopper at the beginning and end of each test and weighing coal added during the test. As a check on weights of coal feed, revolutions of the feed screw are noted and the pounds of coal fed per revolution calculated.

* Continuous record.

TABLE 4.—*Typical Condensed Data Sheet and Analyses of Material Removed*

Test Number 52 | Date July 16–21, 1940
Conditions of Test: Draft 0.02 in. H_2O Excess Air Setting, 30 per cent
Efficiency 12 Hr., 69.5 per cent; 5 days, 58.5 per cent.

	Period										
	Start over night 15/45	60/0[a]	10/50	60/0	20/40	5/55	60/0	30/30	3/57	60/0	Total 5 Days[b]
Length of test, hr.	15.5	12	13.5	2	5	17	2	5	65	1.5	123
Length of On time, hr.: min	5:02	12:00	2:11	2:00	1:40	1:26	2:09	2:30	3:20	1:30	28:37
Coal fed, lb.	77.8	185.8	33.8	30.9	25.9	22.2	30.9	38.7	51.6	23.2	443
Coal per hour (cont. run), lb.											15.48
Coal per hr. (total time), lb.		15.48	2.50	15.48	5.18	1.31	15.48	7.74	0.79	15.48	3.61
Water evaporated, lb.		1639	311	225	228.5	175.5	190	347.5	175	134	3425.5
Water per pound coal fed, lb.		8.82	9.20	7.28	8.81	7.90	6.14	8.89	3.39	5.77	7.72
Average steam pressure, lb. per sq. in.		2.25									
Average output per sq. ft. steam radiation		630	107	520	212	47.8	440	321	12.5	413	129
Flue-gas analysis, average per cent:											
CO_2, on		14.4			10.6			13.0			
CO_2, off					7.6			8.5			
O_2, on		4.8			9.1			6.4			
O_2, off					12.3			11.4			
CO, on		0.0			0.0			0.0			
CO, off					0.3			0.36			
Average draft over fire, in. of water: On		0.020	0.017		0.021	0.018		0.018	0.018	0.023	
Off			0.041		0.044	0.039		0.044	0.037		
Average windbox pressure, in. of water		0.29	0.37		0.36	0.37		0.41	0.60	0.61	
Average fuel-bed resistance, in. of water		0.23	0.31		0.27	0.28		0.32	0.51	0.51	
Average flue-gas temperature (cont.), deg. F		733									
Average maximum flue-gas temperature, On, deg. F			657		712	601		789	482	860	(Max.)
Average minimum flue-gas temperature, Off, deg. F			258		286	246		311	222		
Average retort temperature (cont.), deg. F[c]		530								340	
Average maximum retort temperature, Off, deg. F[c]			511		703	523		493	435		
Average minimum retort temperature, On, deg. F[c]			422		612	433		394	383		
Current, total, watt-hr.		1563	302	261	225	207	263	325	466	192	3804
Average power, watts per hour		130	137	130.5	135	131	131.5	130	144	128	133
Revolutions feed screw: total		411	76	69	58	50	69	87	118	51	989
Per hour		34.0	34.4	34.5	34.8	34.9	34.5	34.7	35.5	34.0	34.6
Average air, cu. ft. per min.: fan		40.5			43.5			43.5			
Over fire		8.0			8.0			8.0			
Excess air (calc.), per cent: On		30.7			76.6			45.3			
Off					135.8			110.5			
Burning rate, lb. per hr., On		14.90			11.72			14.34			

	For Complete 5½ Days
Clinker, lb.	9.83
Loose ash, fire box, lb.	20.25
Coke, fire box, lb.	14.53
Fly ash, boiler tubes, lb.	0.86

[a] 60 min. On, 0 min. Off each hour. [b] Does not include 15.5 hr. starting period. [c] Thermocouple in outside shell of retort (old position, later changed).

TYPICAL ANALYSIS OF MATERIAL REMOVED AT END OF TEST

Clinker:	
Weight, lb.	9.83
Ash, per cent	99.2
Ash fusion, deg. F	2390
Loose ash:	
Weight, lb.	20.25
Ash, per cent	80.9
Ash fusion, deg. F	2460
Coke:	
Weight, lb. (exclusive of amount in loose ash)	14.53
Proximate analysis:	
Volatile matter, per cent	1.2
Fixed carbon, per cent	92.3
Coke:	
Sulphur, per cent	0.59
Ash, per cent	6.5
Apparent specific gravity	0.685
True specific gravity	1.97
Fly ash:	
Weight, lb.	0.86
Volatile matter, per cent	11.0
Ash, per cent	48.8
Screen test:	
Per cent on 100 mesh	27.94
Per cent through 100 mesh	72.06
Weight per cu. ft., lb.	23.10

The coal-feed rate of the stoker in the tests being reported increased gradually as the hours of use increased. When new, the stoker fed 0.420 lb. per revolution of

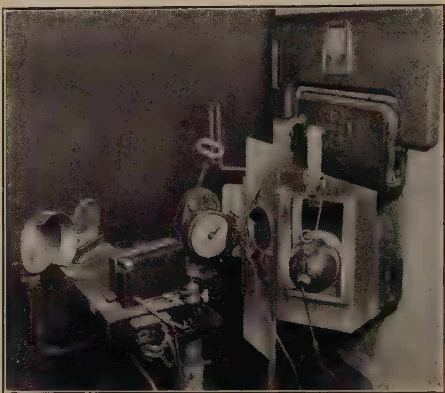

FIG. 9.—ARRANGEMENT FOR PHOTOGRAPHING FUEL BED.

the feed screw. After approximately 15 tons of coal had been burned, the feed rate, using the same coal, had increased to 0.450 lb. per revolution, an increase of 7.1 per cent. This increase agrees with experience in the field, where the rate has been noticed to increase with polishing of the feed screw.

Running Time.—The total running time of the stoker is shown by an electric clock wired in the stoker circuit.

Power.—A recording wattmeter charts power demand. Because of the stoker air control and the size and structure of the coal used, variations in power were slight in the tests reported here. With changing amounts of air from the fan or with coal containing large pieces of hard structure significant variations might be expected.

Total power consumption is measured by an ordinary watt-hour meter stepped up 100 times.

Air Delivered by Fan.—Air delivered by the stoker fan is measured by means of a velometer. The velometer tube is placed in the air duct midway between the air-control damper and the windbox. Since conditions for streamline flow are not met, results are corrected by applying a factor based on excess-air calculations for 12-hr. continuous running periods and over-fire air measurements.

Water Evaporated.—The water evaporated is determined by means of calibrated water tanks from which the water is fed to the boiler through an automatic feed-water regulator.

Steam, after passing through a separator, is discharged from the boiler through a calibrated orifice. A recording steam-pressure gauge gives a continuous record, which can readily be translated into boiler output. Thus the output at any instant is indicated.

Retort Temperature.—Retort temperature is measured by a thermocouple embedded in the inside of the casting of the retort 2¾ in. below the top of the retort.

For some of the earlier tests the thermocouple was in the outside shell but it was found that the temperature at that point was abnormally high at the beginning of the test because of absence of ash on the hearth.

Flue-gas Analysis.—Samples of flue gas are taken at a uniform rate by an automatic sampling device during the 12-hr. continuous run and the two 5-hr. intermittent test periods. Separate samples are taken for each half-hour of the continuous run and for each On and each Off period of the intermittent runs, except that when the Off periods are 40 min. long two 20-min. samples are taken for each Off period.

The CO_2 recorder used throughout the test is checked at intervals by analyzing spot samples with the Orsat apparatus.

Smoke.—Smoke is measured by means of a photoelectric cell and a standard light source mounted on a vertical portion of the smoke pipe near the boiler. This arrangement is not entirely satisfactory because fly ash held in suspension by the gas stream affects the reading and there is no way to check the zero point during a test.

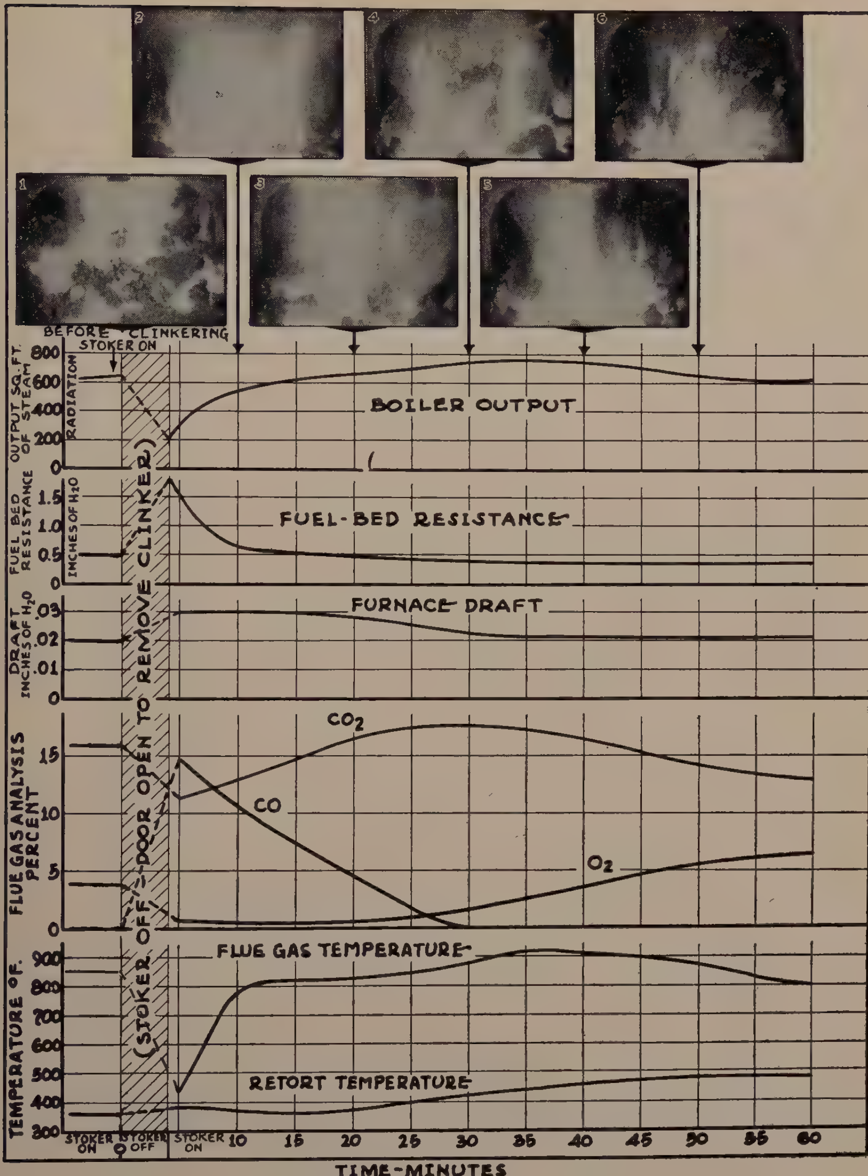

FIG. 10.—EFFECT OF CLINKER REMOVAL.

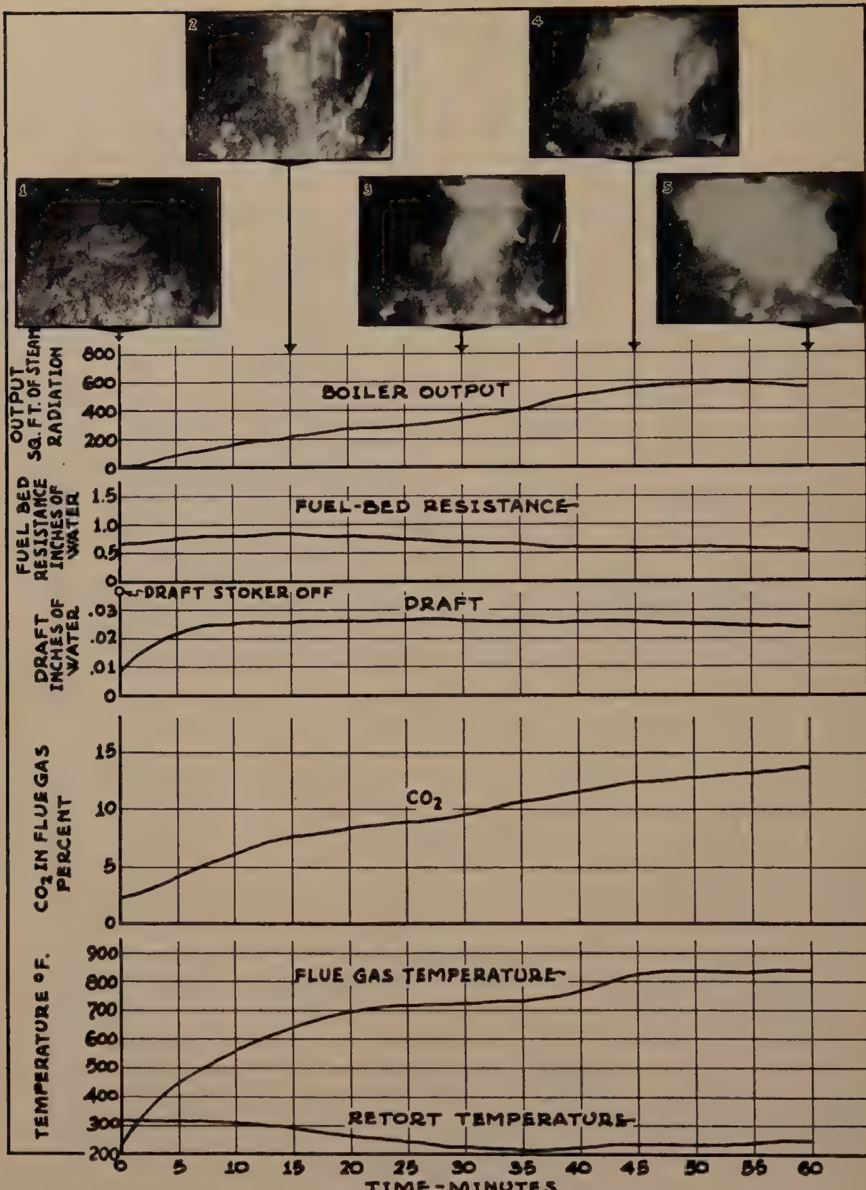

Fig. 11.—Response after hold-fire.

Photographic Record.—A photographic record of the fuel bed is obtained by means of a movie camera and lights, both automatically operated by a time-lapse control. A special fire door was devised to permit the taking of photographs without opening the door. A clock included in each picture by a mirror arrangement aids in correlating the photographs with recorder charts and other data. This setup is shown in Fig. 9.

Clinker Removal.—At the end of the test the fire door is opened to allow the boiler to cool. After 2 hr. the clinkers are removed with tongs. All pieces of clinker under 3 in. in diameter are left to be removed with the loose ash. By this method the material removed as clinker is similar to that removed in practice. The number of pieces removed has varied from three to a maximum of five. Since each test is started with a clean hearth, some of the ash in the coal accumulates at the sides and corners of the firebox as loose ash, which ordinarily is not removed in cleaning a stoker fire. Therefore, the amount of clinker is considerably less than would be obtained for a similar period starting with a normal ash bed.

Fly Ash.—At the end of each test, fly ash is removed from the boiler passes by scraping and brushing. It is carefully collected and weighed and then sent to the laboratory for the tests indicated.

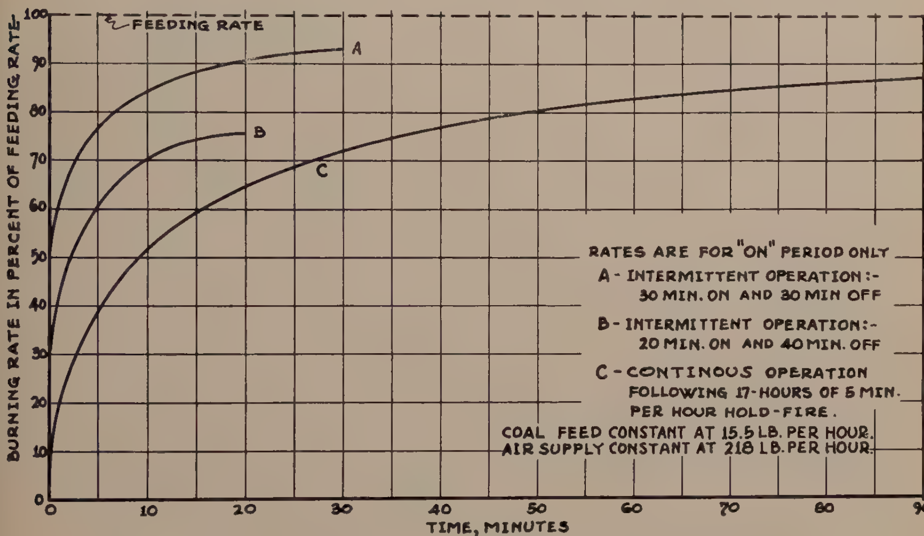

FIG. 12.—BURNING RATES WITH INTERMITTENT OPERATION.

Fuel-bed Resistance.—The resistance of the fuel bed is calculated for On periods only by the following method. The numerical sum of the wind-box pressure and the furnace draft, both in inches of water, gives the total pressure drop through the retort and fuel bed. The retort resistance for the volume of air being delivered by the stoker fan is then read from a chart such as Fig. 5. Subtracting the resistance of the retort from the total pressure drop gives the fuel-bed resistance.

APPEARANCE OF FUEL BED VS. PERFORMANCE

One aim in planning the work was to correlate fuel-bed appearance with performance. This has been done by comparing the movie film with the recorder charts. Two examples of periods in which the fuel bed underwent pronounced changes will be given. The photographs are black and

white enlargements made from the 16-mm. colored movie film, and much of the detail that can be shown on the screen has been lost.

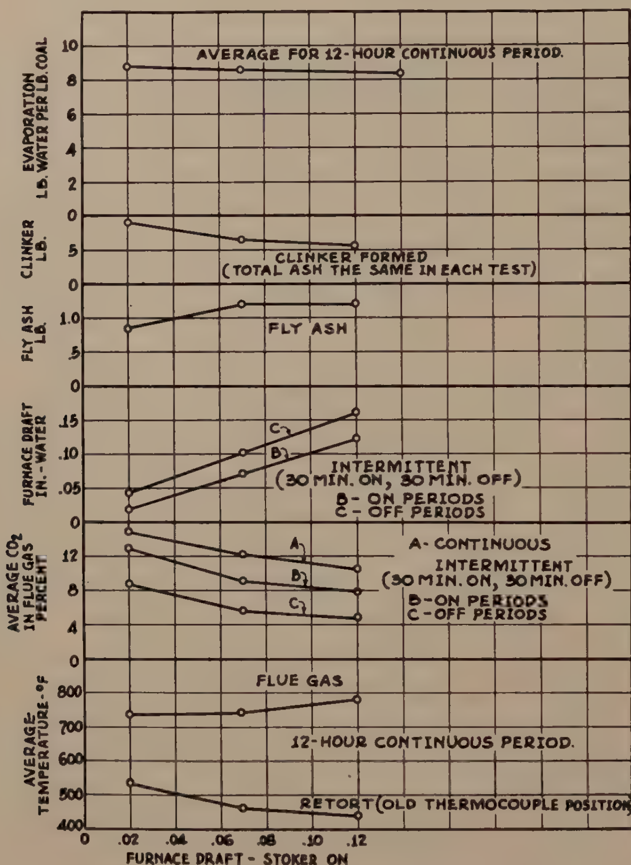

FIG. 13.—EFFECT OF STACK DAMPER SETTING.

Effect of Clinker Removal.—When clinker is removed the fuel bed is drastically disturbed. The effects of removing clinker during a continuous run, which had been started with a normal bed of ashes in the firebox, are shown in Fig. 10. In this test the coal feed was 15.5 lb. per hour and the air supply was constant at 216 lb. per hour. The stoker was Off 4 min. to remove the clinker, which weighed 5.6 pounds.

The most striking immediate effects of clinker removal were the very high resistance of the fuel bed and the high percentage of carbon monoxide (CO) in the flue gases when the stoker was first turned on. Both of these were the result of breaking up the coke in the fuel bed. The leveled bed was much less efficient than the original comparatively ragged bed.

After 15 min. of stoker operation the fuel-bed resistance was lower than it had been before the clinker was removed.

Response after Hold-fire.—During a prolonged hold-fire the fuel bed gradually diminishes in depth and the live fuel bed recedes toward the retort. To check the response after a long hold-fire period the stoker was operated continuously for 1½

hr. after 65 hr. at 3 min. On per hour. The changes that took place during the first hour of one of these "pick-up" periods are shown in Fig. 11. During the hold-fire period the water was kept warm enough so that heat output was recorded after the first few minutes of operation.

The changes in fuel-bed resistance are worth noting. During the 65-hr. hold-fire the resistance of the fuel bed had increased from 0.36 to 0.72 in. of water, even though only 54 lb. of coal had been fed. As Fig. 11 shows, after the first 15 min. of continuous operation following the hold-fire, the fuel-bed resistance decreased as the fuel bed increased in depth.

With this coal (and also with other premium stoker coals tested) there appears to be no definite relationship between the depth of an undisturbed fuel bed and the fuel-bed resistance. Ash and clinker appear to have much more effect on the resistance of the fuel bed.

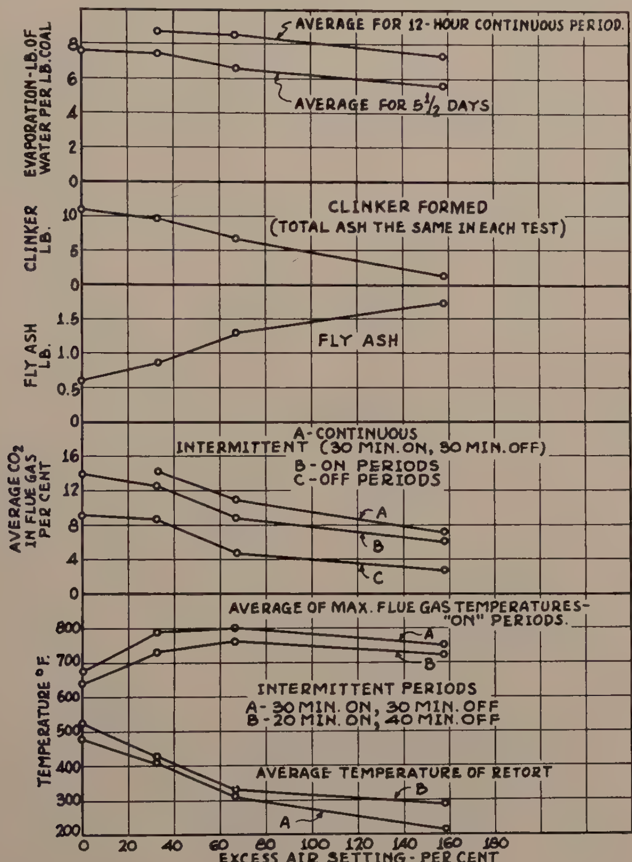

FIG. 14.—EFFECT OF AIR SETTING.

BURNING RATES

With the usual intermittent operation of a domestic stoker, the coal-burning rate when the stoker is running is almost always less than the coal-feeding rate. This fact has been pointed out by various investigators.[2,3]

While no attempt will be made here to go into all the implications of this subject, its importance must not be overlooked in conducting investigations of domestic stokers. It should also be given due consideration in stoker design and application.

One characteristic clearly indicated in these tests is that with a given coal feed and a constant volume of air supply the burning rate when the stoker is running varies widely, depending upon the length of the On periods and the length of the intervals between On periods.

Fig. 12 shows the burning rates during On periods for three different timer settings, all at the same coal feed and air-supply rate. The air supplied was 30 per cent in excess of that theoretically required to burn the coal at the rate it was being fed. Curves *A* and *B* are averages for five On periods.

Effect of Stack-damper Settings

Some of the results of a series of tests in which only the stack-damper setting was varied are presented in Fig. 13. In this series of tests the average excess air for the 12-hr. continuous period was 30 per cent when the draft was 0.02 in. of water. The actual amounts of air supplied with the different damper settings are given in Table 1. The coal-feed rate was 15½ lb. per hour and the total coal fed in each test about 520 lb. Increasing the furnace draft had a pronounced effect on clinkering, the quantity of fly ash, and the CO_2 content of the flue gases. The conclusion reached was that a low draft should be maintained in the furnace at all times.

Effect of Air Setting

Fig. 14 presents some results of a series of tests in which the setting of the automatic air control was varied while furnace draft and coal feed were kept the same. For these tests the coal feed was 15.5 lb. per hour and the stack dampers were set to give a furnace draft of 0.02 in. of water when the stoker was running. "Excess air setting" is the percentage of air delivered, when the stoker was On, over and above the theoretical amount required to burn the coal at the rate it was being fed. This air includes leakage over the fire as well as fan air. During Off periods and On periods of intermittent runs the actual excess air was considerably higher than was indicated by the setting. With an excess air setting of zero, a 12-hr. continuous run could not be made, as the firebox would fill with coke in a short time, owing to the feeding rate exceeding the burning rate. Therefore, for this 12-hr. period, 30 per cent excess air was supplied. Results for the complete test undoubtedly are affected by this deviation.

The higher air settings had the disadvantages of decreased efficiency, increased fly ash, and poorer clinker formation. The lowest setting (supplying the theoretical amount of air) had the disadvantages that some smoke was produced and that the stoker could not be run continuously. An excess air setting of about 30 per cent gave the best results.

Acknowledgments

The assistance of C. H. Sawyer and Harvey Waechter in conducting the tests and of E. J. Maloney in preparing the graphs is gratefully acknowledged.

References

1. R. A. Sherman and E. R. Kaiser: Relation of the Size of Bituminous Coals to their Performance on Small Underfeed Stokers. Bituminous Coal Research Inc., *Tech. Rept.* No. 1, part I, 1935.
2. R. A. Sherman and E. R. Kaiser: Combustion of Bituminous Coal on the Small Underfeed Stoker. *Trans.* A.I.M.E. (1938) **130**, 388.
3. A. P. Kratz and S. Konzo: Performance of a Stoker-Fired Warm-Air Furnace as Affected by Burning Rate and Feed Rate. *Heating, Piping, and Air Conditioning* (Jan. 1940) **12**, 55.

DISCUSSION

(*J. B. Morrow presiding*)

L. A. Shipman,* Knoxville, Tenn.—I want to emphasize especially the fact that the thoroughness of the experiments and the careful analysis given to the results obtained before conclusions

* Southern Coal and Coke Co.

were reached, and the caution of the authors that the facts so presented are true only of the conditions encountered or fixed during the time of the experiments, not generally true, limits the discussion to imagining stimulated by the paper. The fact that this paper so stimulates the curiosity for more information and leads to suggestions and questions that require further research only emphasizes the value of the paper.

The close sizing of the coal as shown by the screen test (Table 4) on top and bottom sizes is responsible no doubt for the most uniform feed of the worm in pounds of coal per hour. This varies only from 15.45 to 15.54 lb. per hour and 0.437 to 0.452 lb. per revolution of the worm.

Practical experience in the field with coal delivered to consumers in which segregation has had its effect show that the coal fed per revolution of the worm varies as to worm, stoker-retort design and coal sizing, and uniform results are not obtained.

It would be interesting to obtain detailed information on the effect of worm design, retort and housing design, and coal bulk density on feeding rates and burning rates, all other things being kept as equal as possible and each varied one at a time. Practical experience has shown that bulk density has a considerable effect on coal feed per revolution of the worm under certain conditions. It is obvious that this test should be set up with as constant a coal feed as possible, to check results of other variables. It would also be interesting to obtain air resistance to different size consists as shown in Fig. 7 at the four quarters of the retort.

Although the summary of burning rates is very short, the findings obtained are of considerable importance. Why does the burning rate vary so much with the different time settings? Is the variation due to reactivity of coal, coal sizing, air carrying the heat away from the fire faster than the fire is producing it but after a certain time interval maintaining an equilibrium?

Such tests as these are the beginnings necessary to obtain the further information needed by Subcommittee No. 2, which has been charged with forming a code for testing coals on underfeed stokers. The authors and their staff are to be congratulated for contributing outstanding information to the meager literature on burning coal in domestic underfeed fuel beds. This work, as well as the work of Sherman and Kaiser of Battelle, Kratz and Konzo of University of Illinois and Barnes of Battelle, will long be used as fundamental textbook material to guide the way to further enlightenment.

W. G. Christy,* Jersey City, N. J.—Many small stokers are sold and installed by dealers who formerly handled radios, refrigerators, and similar equipment. Many coal dealers are now handling stokers. Some of these men are not well posted on proper installation and operation of stokers, and the success of such equipment depends very largely on how well the installation is engineered. Fortunately, engineers are giving this field more attention.

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Control of Coke-tree Formation in Domestic Underfeed Stokers

By C. C. Wright* and T. S. Spicer,† Member A.I.M.E.

(New York Meeting, February 1942)

A characteristic property of bituminous coal is that upon being heated the coal becomes plastic, evolves volatile gases, and finally solidifies into coke. This fundamental characteristic is of utmost importance and is utilized in the production of the high-temperature coke upon which the steel industry is so dependent; but from the standpoint of operation in the domestic underfeed stoker this coking tendency seriously interferes with efficient combustion.

When the coke formed has too strong a structure, certain difficulties result, such as low rate of burning, slow response to heat demand, and, in extreme cases, extinction of fire. The plastic mass prevents uniform distribution of air through the fuel bed and, with strongly coking coals, large dense pieces of coke are produced which have much less reacting surface than was possessed by the original coal. In both ignition and combustion of solid fuels, the extent of the reacting surface is an accelerating factor. Poor combustion permits a surplus of fuel to accumulate in the furnace because the rate of feed and the rate of combustion are not in equilibrium, and this produces an unbalanced condition detrimental to continued automatic service. In order to burn strongly coking coals satisfactorily in a conventional stoker during the three requisite types of operation—continuous, intermittent and hold-fire, frequent attention is necessary.

Domestic stokers are defined by the industry as residential stoker heaters that burn 60 lb. of coal or less per hour. With a few exceptions, these units employ the underfeed principle of combustion in which both air and coal move in the same direction up and through the fuel bed. Domestic combustion equipment has difficulty in meeting the problems presented by the several variations in ash and volatile matter, in ash-clinkering tendencies, in coking properties, and in the particle size of bituminous coals. The work of Bituminous Coal Research Inc. at Battelle Memorial Institute has shown that, of these, coking characteristics and the clinkering tendencies have the greatest effect upon the operation of the conventional clinkering type of stoker.

Typical coke formations in a clinkering type of underfeed domestic stoker are illustrated in Fig. 1, which shows (*A*) the ideal fuel bed obtained with a noncoking or free-burning coal; (*B*) the type of fuel bed formed with a coal that produces a weak coke; and (*C*) the type of fuel bed obtained with a strongly coking coal. *C* is characteristic of the fuel bed developed when the average Pennsylvania bituminous coal is burned in an underfeed stoker of conventional type. It is apparent from these pictures that the formation of coke trees will seriously interfere with the efficient automatic combustion of Pennsylvania bituminous coals.

Contribution from the Department of Fuel Technology of The Pennsylvania State College. Investigation jointly sponsored by the Commonwealth of Pennsylvania, the Western Pennsylvania Coal Operators Association and the Central Pennsylvania Coal Producers Association. Manuscript received at the office of the Institute Dec. 27, 1941. Issued as Contribution 123, August 1942.

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Theory of Coal Oxidation

All coals combine with oxygen at ordinary temperatures if exposed for a sufficient period of time. The sorption of oxygen by the coal begins as soon as it is broken out of the bed, and is most rapid with freshly mined coal. The most important factors affecting the oxidation of coal are: (1) time of treatment, (2) temperature, (3) moisture, (4) size of coal, and (5) ratio of volume of oxygen to weight of coal being oxidized.

Fig. 2[1] shows the increase of oxidation of dry coal with increase in temperature and illustrates the marked effect of temperature upon the reaction. Haslam and Russell[2] state that the oxidation and spontaneous combustion of coal occurs in five more or less distinct stages:

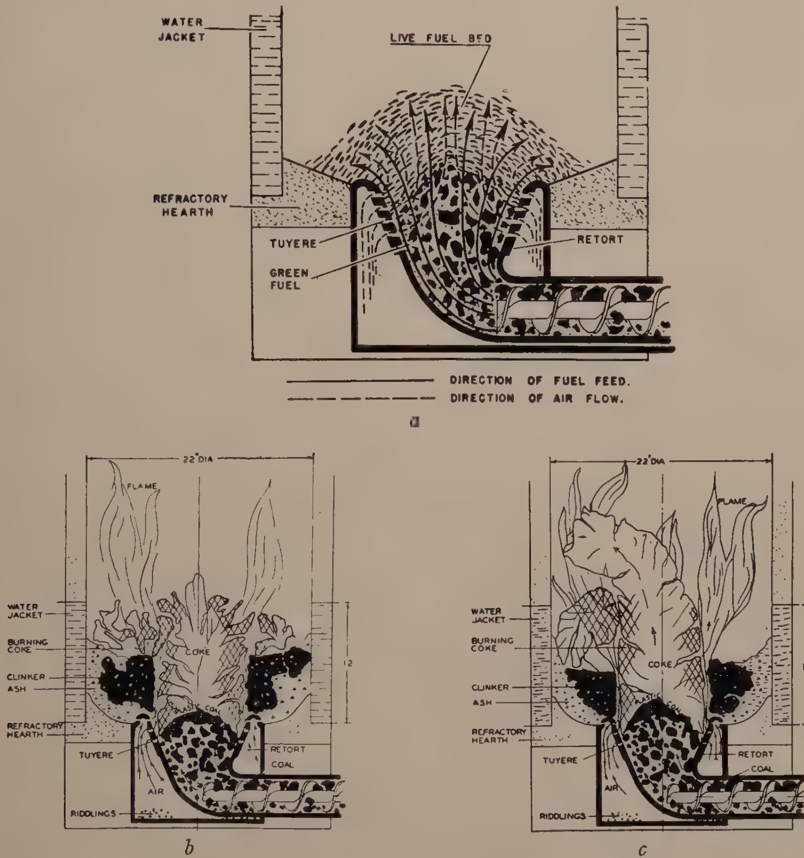


Fig. 1.—Typical fuel beds in an underfeed stoker.

From publications by Battelle Memorial Institute.[6,7] *a*, ideal bed; noncaking coal. *b*, weakly caking coal. *c*, strongly caking coal.

At first the coal begins to absorb oxygen slowly until a temperature of 120°F. is reached. The second stage begins at 120°F. and more rapid absorption of oxygen continues until the coal has reached 175° to 280°F. The temperature depends upon the quality of coal, on its fineness of division, and other factors. The third stage begins at this latter temperature and differs from the second stage in that CO_2 and water vapor are given

[1] References are at the end of the paper.

off as the absorption of oxygen accelerates. This rapid absorption of oxygen and liberation of CO_2 and water vapor continue to a temperature of about 450°F. at which temperature

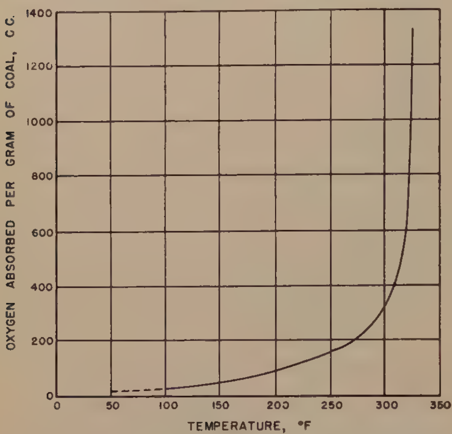


FIG. 2.—RELATIVE RATE OF OXIDATION OF DRY COAL AT VARIOUS TEMPERATURES. (FROM S. W. PARR.[1])

the fourth stage begins. In this stage the oxidation is spontaneous (i.e., does not require heat from an external source) and the liberation of water vapor and CO_2 is more rapid. The fifth stage, namely, the actual combustion or kindling of coal takes place at about 660°F. Coal then continues to burn vigorously as long as air is supplied.

It is well known that weathering or slow oxidation at atmospheric conditions markedly affects the appearance, size, heating value, firing, and coking qualities of many coals. Coking qualities are especially sensitive, so that a few months weathering may seriously decrease coking quality, and it is virtually impossible to obtain a good coke from coal that has been stored in the open for more than two years. The coking qualities of coals may also be impaired seriously by exposure to oxygen at slightly elevated temperatures for any appreciable periods. The Bureau of Mines[3] reports that mild oxidation of the Pittsburgh bed coal at 86°F. is accompanied by a progressive change in carbonizing properties even before it affects significantly such properties as agglutinating value, chemical analysis, and heating value.

For many years it was generally believed that the change in coking properties of bituminous coals due to oxidation were solely of an undesirable character. More recently, however, it has been established[3,4,5] that some high-volatile coals that developed a high degree of fluidity during carbonization show improved coking characteristics after oxidation has progressed to a certain stage peculiar to each coal. It has been found of economic advantage to reduce the fluidity of such coals so that they will produce more desirable products when subjected to low-temperature carbonization. Moreover, it has been noticed that when part of the coal seam outcrops and is thus exposed to mild atmospheric oxidation, the portion of the coal near the surface does not coke as strongly as coal from a deeply mined portion of the same bed.

With the foregoing fundamental knowledge as a basis, it has been the aim of the authors to utilize this information in the development of an improved bituminous stoker. It was believed that with any coking coal, by proper mechanical design, rapid preoxidation and modification of coking characteristics could be attained. This would eliminate one of the most serious problems confronting the domestic-stoker industry. As will be borne out by the experimental data herein presented, the desired result has been accomplished by simple modifications of existing equipment.

DEVELOPMENT OF EQUIPMENT FOR CONTROL OF COKE-TREE FORMATION

A field survey of existing stokers was made through contacts with home owners, stoker dealers, stoker manufacturers, coal dealers, coal producers, and research laboratories to determine what existing units offered possibilities for use with Pennsylvania coals. As a result of this survey, several stokers of favorable design were

purchased for testing purposes in the laboratories. Five units of different designs, but all of the general underfeed type, were tested with strongly coking Pennsylvania bituminous coals to ascertain primarily whether they could burn this class of coal satisfactorily. The results showed that none of these stokers would handle with satisfaction the Pennsylvania coals tested.

It was found that one of these units could be adapted for studying the effect of preoxidation on the combustion of coal, and in the early stages of the project development work was restricted to this unit. One of the features of this unit is the vertical screw and it was believed that, aside from the reduction of segregation and degradation derived from such a screw, the coal would be so agitated that the air would come in contact with most of the coal before a material rise occurred in its temperature. A diagrammatic sketch of the original stoker unit is shown in Fig. 3.

Modification of Stoker A

The first attempt to preoxidize coal in this stoker was made by introducing compressed air through the "smoke-back" connection shown in Fig. 3. Tests with this modified unit gave remarkable results. By adjusting the compressed air so that the right percentage of the total air required for combustion would be delivered for the preoxidation of the coal, a bed could be maintained that would not develop coke trees. Other improvements observed were higher CO_2 content of the stack gases (10 to 15 per cent), faster heating response, and more dependable hold-fire control. One serious objection to this first modification was that because the air entered at only one point the rotation of the vertical screw produced a surging action. Moreover, because about 10 to 35 per cent of the air was induced through a ½-in. compressed-air line, the high velocity of these surges of air caused an excessive suspension of fly ash. The coals tested with this modified stoker were from the Pittsburgh seam, the Lower Kittanning seam, and the Upper Freeport seam. Observers of these combustion tests agreed that a definite improvement in performance had been attained.

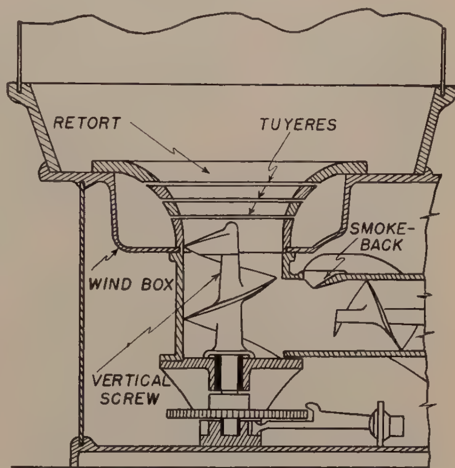


Fig. 3.—Retort section of original stoker.

In the next stage of development compressed air was delivered at three other points beside the "smoke-back"; that is, in the side of the vertical screw housing and below the top of the screw. The purpose of this change was to reduce the initial entrance velocity of the preoxidation air and to distribute this source of air more evenly in the hope that less fly-ash would be thrown into suspension and localized overheating eliminated. Tests were then run on several coals with this second modification and a general improvement noted Fig. 4 shows photographs of a typical fuel bed when burning a strongly coking coal with admission of preoxidation air (*a* and *b*); also the same bed a short time later after admission of preoxidation air had been discontinued but total air supply maintained the same (*c* and *d*). It should be noted that the use of a vertical screw without preoxidation air has little, if any, effect in eliminating coke trees.

Despite the improved results obtained with the preoxidation principle utilizing

compressed air, the question "Is it practical?" had yet to be answered. To be fitted for practical operation, the stoker must operate as a self-contained unit using variable speed control. A new four-point manifold was designed and installed together with a proportioning chamber, which provides a flexible means of proportion-

FIG. 4.—FUEL BEDS IN STOKER A. LOWER KITTANNING SEAM COAL.

a, stoker "on." Operation consisted of 1 hour continuous followed by 20 minutes hold-fire and then 20 minutes "on." Preoxidizing air used. 15 per cent CO_2 at 960°F.

b, taken 5 minutes after picture *a*. Stoker "off." 8 per cent CO_2, stack temperature falling.

c, "off" period shown in *b* lasted for 33 minutes, after which stoker was started *without* preoxidizing air. *c* was taken after 28-minute continuous operation. 2 per cent CO_2, 500°F. stack.

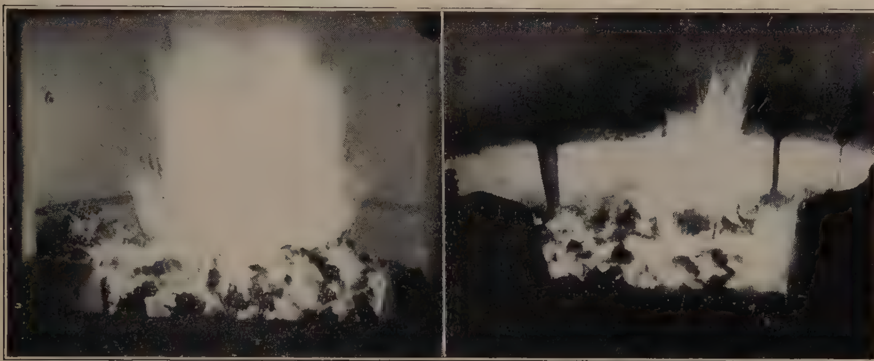
d, taken after 39-minute continuous operation (11 minutes after *c*) *without* preoxidizing air. Note rapid formation of coke tree. 3 per cent CO_2, 450°F. stack.

a single fan to supply air for both the preoxidation process and the regular tuyeres above the vertical screw. Measurements were made of the resistance of the column of coal plus the resistance of the fuel bed in order to determine the amount of pressure against which a fan must operate. Back pressures recorded ranged from 0.6 to 0.8 in. of water with a very thick bed of coal and coke. The fan supplied with this stoker was not capable of efficient operation against even this low resistance and it was replaced by a blower having a ing the air between the lower preoxidation tuyere and the combustion tuyeres above it.

As a self-contained unit rather than one utilizing an outside source of compressed air for preoxidation, this modified stoker was still able to handle all the strongly coking coals of Pennsylvania. Thus, the general theory that preoxidation could be used as a means of eliminating excessive coke trees appeared to be a practical proposition. Fig. 5 shows pictures of typical fuel beds of coal from the Upper Freeport seam

when burned with the aid of the self-contained unit.

The unit was next modified to eliminate the use of a manifold for the induction of preoxidation air and to substitute a more practical method, a lower tuyere. The stoker thus modified was then used on a wide variety of coals for several weeks at a time, to heat the stoker laboratory in accordance with the demands of the room thermostat. During this time it gave excellent performance and the fire did not go out even when a hold-fire setting as low as 0.5 lb. per hour was employed. At the end of the test the stoker was dismantled, and it was observed that the wind box contained virtually no riddlings.



a *b*

FIG. 5.—FUEL BEDS IN STOKER A AFTER CONVERSION TO SELF-CONTAINED UNIT. UPPER FREEPORT COAL.

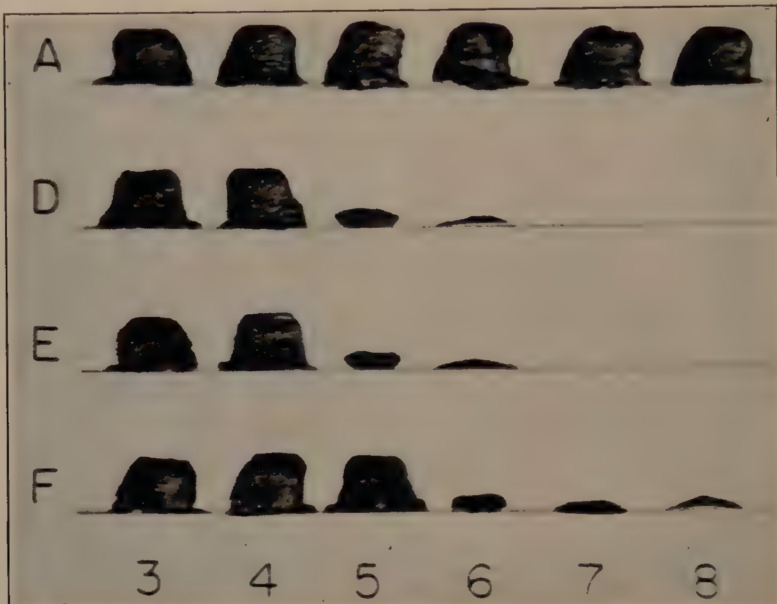
a, taken after 3 hours 15 minutes continuous operation; 10 lb. per hour feed rate.
b, taken during "on" period shortly after stoker had been "off" for 1½ hours; 20 lb. per hour feed rate.

MECHANISM OF PREOXIDATION

Although the modifications described in the preceding section produced results in agreement with predictions based on theoretical considerations, it appeared desirable to determine whether the results were due to predicted causes or to some fortuitous combination of circumstances. The retort of the modified stoker, therefore, was drilled to provide eight sampling stations equally spaced between the bottom of the vertical screw and the level of the upper regular tuyere. During a series of test runs with coals from the Upper Freeport, Lower Kittanning, and Pittsburgh seams, the temperature of the coal in the retort at each station was measured and samples of coal were extracted. The samples removed were then ground individually to minus 60 mesh and each subjected to the British standard crucible swelling test.

In all cases, where no preoxidizing air was admitted, the swelling numbers exhibited by these samples from stations 1 to 8, inclusive, showed the same value as the original coal and in all cases coke trees were present in the fuel bed. Using preoxidizing air with Upper Freeport ½-in. to 0 slack, the results shown by pictures and data in Fig. 6 were secured indicating an abrupt change in coking properties of the coal about 1 to 1½ in. below the upper regular tuyere. Similar results were obtained with Lower Kittanning ¾-in. to 0 slack. Using Pittsburgh-seam double-screened coal, 1¼ to ⅜-in., however, no change in swelling number of the coal from the various sample ports could be detected despite the fact that the improvement in fuel-bed conditions was pronounced. Failure to detect a change in swelling number with this coal may be due to the difficulty

of sampling this larger coal, to the fact that the oxidation occurred at a much higher level, or more likely, to the fact that the oxidation is primarily a surface phenomenon. With slack coal the surface is large per unit of volume, while with the double-screened coal the surface is comparatively small; hence, with the sized coal sufficient surface oxidation may have occurred to prevent coalescence of the particles, yet not sufficient to materially affect the over-all swelling properties of the coal after it has been ground to 60 mesh.



Sampling port number

FIG. 6.—EFFECT OF PREOXIDATION ON SWELLING PROPERTIES OF UPPER FREEPORT ½-INCH TO 0 SLACK COAL.

Conditions of Tests

Series	Type of Operation	Rate of Feed, Lb. per Hr.	Air Entering	Condition of Bed
A	Continuous	10	Tuyeres only	Medium size coke tree
D	Continuous	20	Manifold and tuyeres	Level bed, no coke tree
E	Continuous	20	Manifold and tuyeres	Level bed, no coke tree
F	Continuous	20	Manifold and tuyeres	No coke tree, level bed 4 in. above tuyeres

Data for temperatures taken at stations 1 to 6, inclusive, are shown in Table 1 for a test with Lower Kittanning ¾-in. to 0 slack with the stoker operating on a 45-min. "on," 15-min. "off" cycle. The measurements were made near the end of the third 45-min. "on" period, when the bed was relatively thin—1 to 2 in. above the regular tuyere. Similar data for a test with Pittsburgh-seam 1¼ to ⅜-in. coal with the stoker on continuous operation are shown in Table 1. The measurement was made after one hour of continuous operation and the fuel bed was level and 3 to 4 in. above the regular tuyere.

It is to be noted that port 6, where the maximum temperature so far recorded is

350°C. (662°F.), is substantially the level of the top of the vertical screw. This temperature is insufficient to damage even ordinary cast iron, which explains why no screw tips have been burned off.

TABLE 1.—*Temperatures in Retort*

Sample Station No.	Temperature, Deg. C.	
	Near Center of Retort	Near Outside of Retort
LOWER KITTANNING ¾-INCH TO 0 SLACK		
1	36	28
2	54	40
3	66	47
4	52	45
5	180	95
6	350	190
PITTSBURGH 1¼- TO ⅜-INCH COAL		
1	36	30
2	44	38
3	64	50
4	50	42
5	110	78
6	110	76

Though the foregoing experiments may not be conclusive evidence of the principle of preoxidation, in combination with performance data they provide strong support for the belief that the theory is correct.

MODIFICATION OF STOKER B

In view of the favorable results secured with modification of stoker A, the work was extended to include the application of the preoxidation principle to another stoker. The second unit selected for conversion was a completely automatic soft-coal stoker incorporating bin feed and ash removal. Previous test work on this unit had demonstrated that, owing primarily to excessive coke formation, it could not burn satisfactorily the strongly coking coal so typical of most of the normal Pennsylvania production.

Modification of this unit was first attempted solely through addition of a preoxidation tuyere, because the method of feeding coal into the retort already provided a mild agitation. This agitation and the mixing of coal and air were found to be insufficient, however, and no improvement was noted in performance of the stoker. This result probably can be attributed to the channeling of the air through the fuel bed.

The unit was further modified to incorporate a vertical screw driven through suitable gears from the regular drive shaft. As a preliminary test, the preoxidation tuyere was sealed off and a number of coals tested to determine the effect of the screw alone. Fuel-bed conditions were slightly improved under certain conditions of operation, but in general the performance was unsatisfactory. After the effect of vertical screw alone had been established, the seal was removed from the preoxidation tuyere and a number of strongly coking coals tested in the modified unit incorporating both the screw and preoxidation air. Prior to modification, much difficulty was experienced in maintaining a fire with these coals, because of excessive formation of coke trees, which impaired efficiency and in several instances extinguished the fire. After modification, satisfactory fuel beds were obtained in which there was little or no accumulation of coke and the fuel burned at a rate nearly equal to that at which it was fed. Fig. 7 shows photographs of typical fuel beds burning strongly coking coals in the original unit and in the fully modified unit.

MODIFICATION OF STOKER C

In order to complete studies on the modification of typical bituminous coal stokers, a third unit—one of the popular clinkering models—was modified to incorporate both vertical screw and preoxidation tuyeres. Combustion performance with strongly coking coals in this unit prior to modification was appreciably poorer than in either of the two units previously modified. After modification, coke trees were completely eliminated and combustion performance was as satisfactory as with the

TABLE 2.—*Description of Coals Tested*

Source			Sample Description, Inches	Moisture As Recd., Per Cent	Proximate Analysis and Sulphur (Dry Basis)				Ash Softening Temperature, Deg. F.	British Swelling Number
State	County	Bed			Volatile Matter	Fixed Carbon	Ash	Sulphur		
Pennsylvania	Indiana	Upper Freeport	½ to 0 slack	1.1	30.8	62.0	7.2	1.59	2590	8.0
Pennsylvania	Indiana	Upper Freeport	½ to ¼	0.9	31.0	61.4	7.6	1.94	2490	
Pennsylvania	Cambria	Upper Freeport	¾ to 0 slack	1.2	25.7	67.4	6.9	1.56	2160	9.0
Pennsylvania	Allegheny	Thick Freeport	⅜ to 0 slack (raw)	1.7	32.4	56.4	11.2	1.93	2180	
Pennsylvania	Allegheny	Thick Freeport	1¼ to ⅜ (washed)	1.0	35.7	57.1	7.2	1.67	2170	8.0
Pennsylvania	Allegheny	Thick Freeport	¾ to 0 slack	1.5	33.3	54.4	12.3	2.38	2180	
Pennsylvania	Clearfield	Lower Freeport	¾ to 0 slack	1.1	23.0	70.2	6.8	0.60	2900+	9.0
Pennsylvania	Cambria	Lower Freeport	¾ to 0 slack	1.0	24.7	68.0	7.3	1.94	2345	
Pennsylvania	Somerset	Upper Kittanning	¾ to 0 slack (O.T.)	0.6	18.3	73.7	8.0	2.07	2290	8.5
Pennsylvania	Clearfield	Lower Kittanning	¾ to 0 slack	1.1	22.0	69.2	8.8	3.03	2310	9.+
Pennsylvania	Clearfield	Lower Kittanning	¾ to 0 slack	1.8	23.7	68.5	7.8	1.21	2590	9.0
Pennsylvania	Clearfield	Lower Kittanning	¾ to 0 slack	0.7	21.8	70.8	7.4	1.37	2620	9.0
Pennsylvania	Centre	Lower Kittanning	¾ to 0 slack ($CaCl_2$)	4.2	27.7	63.3	9.0	1.10	2800	
Pennsylvania	Cambria	Lower Kittanning	1¼ to ¼	1.0	18.3	77.0	4.7	1.47	2430	9.0
Pennsylvania	Cambria	Lower Kittanning	1¼ to ¼	0.8	21.4	73.8	4.8	1.35	2400	9.+
Pennsylvania	Cambria	Lower Kittanning	¾ to ⅜ (O.T.)	0.4	22.8	70.6	6.6	1.64	2670	9.+
Pennsylvania	Allegheny	Pittsburgh	1¼ to ⅜	1.9	37.2	54.7	8.1	1.81	2500	
Pennsylvania	Washington	Pittsburgh	1⅛ to 0 slack	2.1	35.8	53.6	10.6	1.82	2280	8.0
Pennsylvania	Allegheny	Pittsburgh	⅜ to No. 10M (heat dried)	1.2	37.2	54.9	7.9	1.97	2620	7.5
Pennsylvania	Allegheny	Pittsburgh	Minus No. 8M (roasted)	1.4	32.8	59.9	7.3	2.27	2020	3.5
West Virginia	McDowell	Pocahontas No. 3	1 to 0 slack	0.7	24.0	72.9	3.1	0.54	2010	8.5
Pennsylvania	Fulton,	Fulton & Barnett	1 to 0 slack (O.T.)	1.4	16.5	76.8	6.7	0.64	2750	8.5
Pennsylvania	Bedford	Fulton & Barnett	⅜ to 0 slack (O.T.)	0.4	16.4	75.9	7.7	1.42	2730	9.0
Pennsylvania	and	Fulton & Barnett	1 to ⅜ (O.T.)	0.3	16.2	76.8	7.0	1.12	2800+	8.5
Pennsylvania	Huntingdon	Fulton & Barnett	1½ to 0 slack (O.T.)	0.3	16.0	76.5	7.5	1.19	2800+	

other two modified units. Although comparative data could not be secured of clinker formation, owing to the inability of the original unit to perform satisfactorily

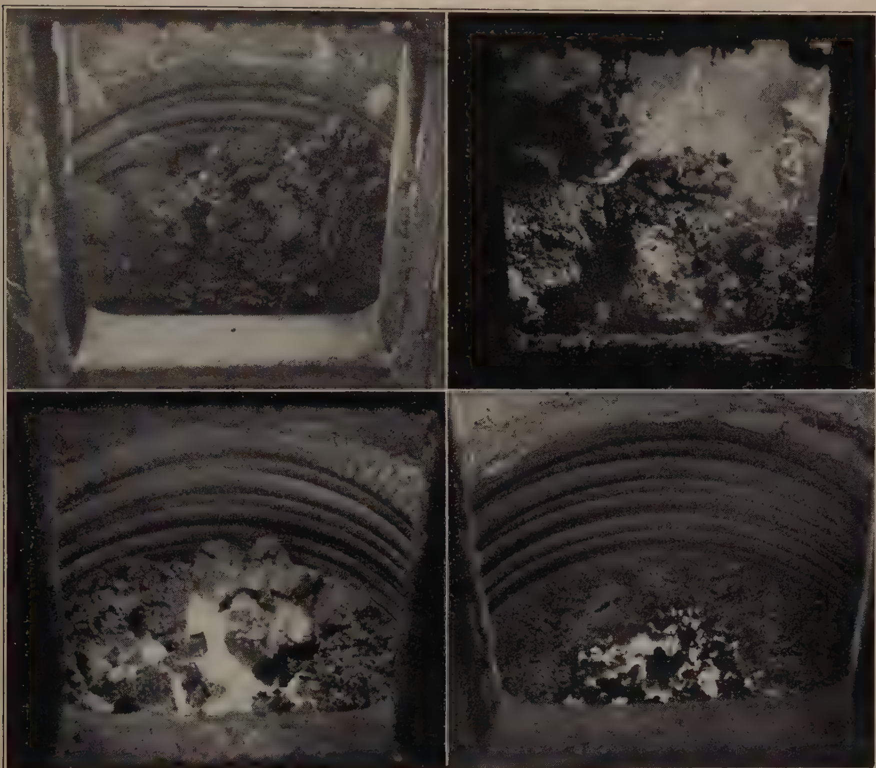


FIG. 7.—FUEL BEDS IN STOKER B BEFORE (*a* AND *b*) AND AFTER MODIFICATION (*c* AND *d*).
a, Upper Freeport seam coal. Fire put out by coke formation and green coal.
b, Pittsburgh seam coal showing excessive coke formation resulting in very inefficient combustion. Taken during "off" period immediately following operation on medium-heavy heating cycle.
c, Upper Freeport seam coal after modification. Taken during "off" period when operating on a medium hold-fire setting.
d, Pittsburgh seam coal after modification showing type of coke produced. Taken during "off" period immediately following operation on a medium heating cycle.

when burning strongly coking coals, no difficulty was experienced in securing an excellent clinker with the coals tested, which ranged in ash-softening temperature from 2180° to 2620°F.

Modification of Stoker B by Independent Organization

On the basis of the excellent performance secured with the modified stoker B in the Comfort Heating Laboratory, one of the sponsoring organizations has undertaken an independent test of the practicality of the modifications. The model used for modification was different from that secured for laboratory test, therefore the details of the changes had to be modified somewhat. However, substantially the same design with minor improvements has been employed. Owing primarily to improved design of the preoxidizing tuyere, and to the fact that double-screened coal has been used instead of the slack supplied for laboratory testing, the performance of this unit has surpassed

that obtained in the laboratory. An indication of the type of fuel bed obtained when a coking coal is burned in this unit is given in Fig. 8, as well as a graphic picture of the rapidity of response to thermostatic demand.

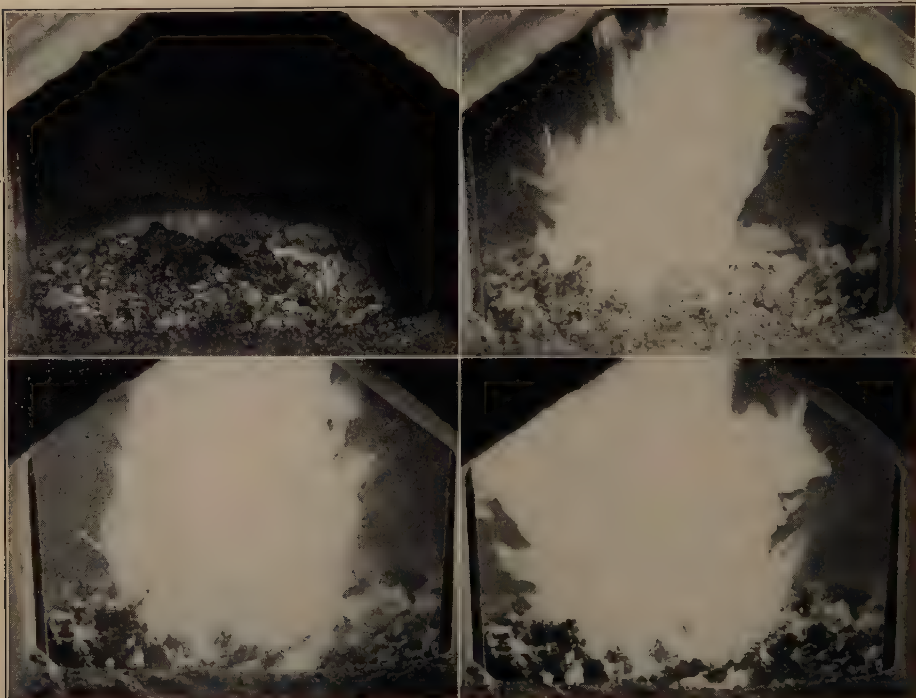


FIG. 8.—FUEL BEDS IN MODIFIED STOKER B TAKEN AT 15-SECOND INTERVALS. UPPER FREEPORT ¼ TO ¾-INCH COAL.
a, just before stoker cut on. *c*, 30 seconds after picture *a*.
b, 15 seconds after picture *a*. *d*, 45 seconds after picture *a*.

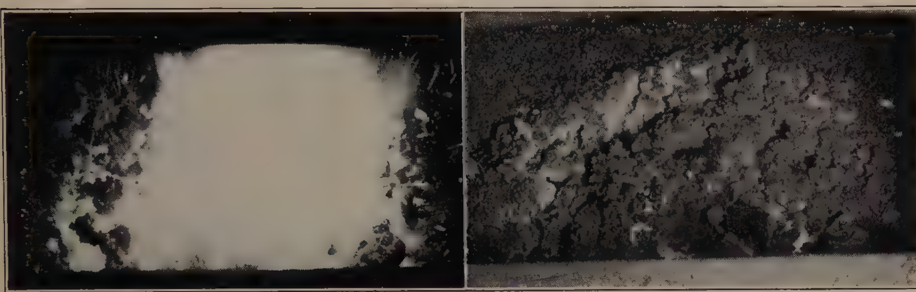
HOME TEST OF MODIFIED STOKER

A regular model of stoker B installed in a State College home has been modified to incorporate the vertical screw and preoxidation tuyere in a manner similar to that employed for the laboratory test model. This modified unit has operated for approximately three months on strongly coking Lower Kittanning coal, with complete satisfaction. Excellent fuel beds are in evidence at all times and the variety of heat demand has been sufficient to indicate that satisfactory performance may be expected in year-round operation. Fig. 9 shows typical photographs of the fuel beds obtained in this unit. No mechanical difficulties have been encountered with either tuyere or vertical screw.

WIND-BOX PRESSURES

In order to supply the air required for preoxidation, it might be assumed that excessive wind-box pressures would be required, but experimental data on the wind-box pressures developed in all three of the different makes of stokers converted to incorporate the vertical screw and preoxidizing tuyeres show that the pressure may be expected to range from about 0.6

to 1.2 in. of water. This is the range of pressure normally encountered in domestic stokers. When preoxidation air is used the wind-box pressure will depend upon the pressure. During this three-month period, the proportioning of air was varied between an estimated 15 and 30 per cent to the preoxidizing tuyere.



a *b*

FIG. 9.—FUEL BED IN MODIFIED STOKER INSTALLED IN HOME. LOWER KITTANNING SEAM COAL. *a*, taken during "on" period. *b*, taken a few seconds after end of "on" period.

relative resistance of preoxidizing tuyere, the primary tuyere, the fuel-bed resistance and the resistance of the column of raw coal above the preoxidizing tuyere, and upon the relative proportion of the air supplied to the two types of tuyeres. As a result of the improved condition of the fuel bed when operating with preoxidizing air, fuel-bed resistance usually is decreased sufficiently to compensate for the increased resistance introduced by the preoxidizing tuyere and the column of raw coal.

As an illustration of practical results on wind-box pressures, the experimental data for home tests of stoker B may be cited. Prior to modification of this stoker, the unit was tested for a period of almost two months in burning a strongly coking Lower Kittanning coal. Daily records of wind-box pressure were obtained and showed variations in pressure between 0.9 and 1.2 in. of water. With no changes to the furnace, in the coal used, or to the stoker other than incorporation of the preoxidation tuyere and vertical screw, the modified unit was operated for a period of three months, and daily records showed a range of wind-box pressures from 0.6 to 0.8 in. of water; an average decrease of 0.3 in. in wind-box

COMBUSTION TESTS IN MODIFIED STOKERS

Combustion tests have been completed on coals from 18 different mines and 8 different seams. In several instances more than one size of coal from the same mine has been tested to determine the effect of size upon performance. In Table 2 detailed descriptions of the coals tested are presented.

Four different types of furnaces have been used, including regular commercial hot-water and hot-air furnaces and the laboratory-designed hot-water furnace. No significant difference has been noted in the performance of the stoker when operating in the different kinds of furnaces, and in all cases satisfactory elimination of coke-tree formation has been secured, and improved combustion obtained.

CONCLUSIONS

Modification of the coking properties of strongly coking coals can be obtained in domestic underfeed stokers by intimate mixing of preoxidation air and green coal in the retort of the stoker when the retort is operating at its normal temperature.

Three different units, representing stokers with hand-operated dry ash removal, auto-

matic dry ash removal, and clinkering ash removal, have been modified in the laboratory to incorporate a vertical screw and preoxidation tuyere. Each of the modified units has performed satisfactorily with complete elimination of coke trees while burning strongly coking coals in the laboratory-designed hot-water furnace and in standard commercial hot-air and hot-water furnaces.

Wind-box pressures on stokers incorporating the preoxidation principle have been found to average between 0.6 and 1.2 in. of water pressure. The decrease in fuel-bed resistance as a result of elimination of coke-tree formation is usually sufficient to compensate for the increase in resistance resulting from the preoxidizing tuyere and the column of raw coal above this tuyere.

Experimental evidence based on coke-button studies of coal extracted from various parts of the retort indicate that modification of coking properties is accomplished an inch or more below the zone of active combustion and at a temperature as low as 300°F.

Application of the modifications to a stoker operating in a regular home installation, and to another stoker by an independent organization, have demonstrated that the modifications are practical. Improved combustion efficiency and response to heat demand has been obtained and hold-fire performance improved in both modified units.

Data on fly-ash and clinker formation are inconclusive at present, but preliminary results indicate that the former may be slightly worse and the latter somewhat improved. Since both these factors are influenced more by retort design and by methods employed for removing the ash from the combustion zone and region of high gas velocity than by the modifications incorporated, it is felt that improvements along these lines are subject to further research.

Acknowledgment

This investigation was made possible through enactment by the 1939 General Assembly of the Commonwealth of Pennsylvania of Act 392 appropriating funds to the School of Mineral Industries of The Pennsylvania State College for a cooperative research program on new and improved uses for coal. The program has been carried out under the supervision of the Department of Mines of the Commonwealth of Pennsylvania and has been financed jointly by funds supplied in equal amounts by the Commonwealth and by industry. The program on bituminous coal has been supported jointly by the Western Pennsylvania Coal Operators Association, the Central Pennsylvania Coal Producers Association and the leading coal carriers, the Baltimore and Ohio, the New York Central and the Pennsylvania Railroads. The problems for investigation have been selected and supervised by a research advisory committee with representation from the coal associations, labor and the Commonwealth of Pennsylvania.

The authors are deeply indebted to the advisory committee for constructive criticism and active assistance through the course of the investigation, to the various coal companies and individuals who supplied coal for test purposes, and to their colleagues of the Fuel Technology Department for assistance and encouragement throughout the course of these studies.

References

1. S. W. Parr: Industrial Coal. New York, 1924. Ronald Press Co.
2. R. T. Haslam and R. P. Russell: Fuels and Their Combustion. New York, 1926. McGraw-Hill Book Co.
3. L. D. Schmidt, J. L. Elder and J. D. Davis: *Ind. and Eng. Chem.* (1936) **28**, 1346–1353.
4. H. J. Rose and J. J. S. Sebastian: *Trans.* A.I.M.E. (1930) **88**, 556.
5. L. D. Schmidt and J. L. Elder: *Ind. and Eng. Chem.* (1940) **32**, 548.
6. C. A. Barnes: Tech. Rept. No. 4, Bit. Coal Res. Inc. (1938).
7. R. A. Sherman and E. R. Kaiser: *Trans.* A.I.M.E. (1938) **130**, 388.

Fuel Technology—Curriculum and Career

By A. W. Gauger,* Member A.I.M.E.

(New York Meeting, February 1942)

It is with some trepidation that I approach my subject, for I know that I shall at once incur the suspicion of the mechanical engineer, with his concern for boiler tests and efficiencies; of the mining engineer, with his mechanization problems; and of the chemical engineer with his unit processes. Nevertheless, it must be recognized that the technology of fuels, the systematic body of knowledge relating to the production, processing and utilization of mineral fuels, has grown to such proportions and involves so many widely diverse applications of fundamental science and engineering that it behooves us to stop and consider just what its status is in relation to other technologies, as well as in relation to the present and future needs of the social order. The time has come when technologists, applied scientists and engineers must consider the possible effects that their achievements may have on Society. The development of a machine to replace several hundred men is a fine achievement when considered solely from the point of view of creative endeavor. But what of the men who are displaced by the machine! Does not the responsibility for the development of this machine embrace as well the consequences of obsolescence and technological unemployment resulting from such development? I believe that it does and that we must recognize this fact in our practice of engineering and technology as well as in the training of future engineers and technologists.

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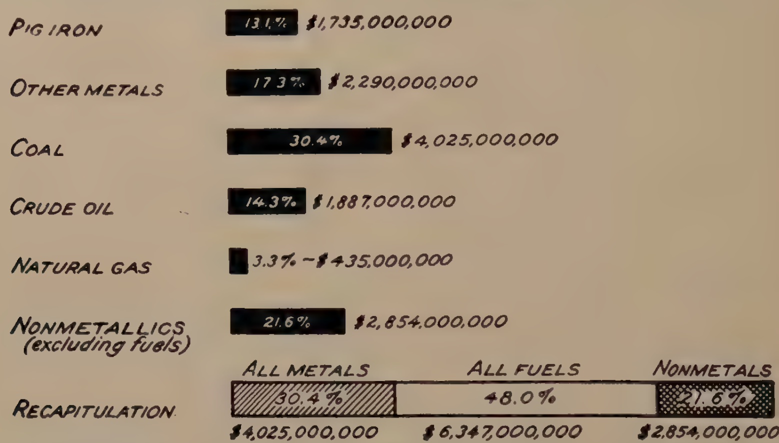
Magnitude of the Fuels Industry

Fuel technology is the body of knowledge that relates to the mineral fuels. The fuels industry is by far the most important of all our mineral industries. The chart in Fig. 1, reprinted from U. S. Bureau of Mines Information Circular No. 6643, indicates the comparative values of metals, fuels and other nonmetallic minerals for the year 1929. Of the grand total of 13¼ billions of dollars in value of minerals at the mines, the mineral fuels contributed 48 per cent. Coal, with 34.4 per cent, exceeded the value of any other single mineral. In fact, the value of the production of the Pennsylvania anthracite mines alone annually exceeds the value of the gold production of the entire world. Petroleum and natural gas in 1929 contributed 14.3 per cent and 3.3 per cent, respectively, of the total value of world mineral production.

So much for the world picture; let us now look at the situation in the United States. Over the 60-year period 1880 to 1939, $46,409,727,000 of metallic products, $80,788,852,000 of fuels (coal, petroleum, natural gas, natural gasoline) and $27,871,000,000 of nonmetallic minerals other than fuels were produced. The mineral fuels have contributed 52.7 per cent of the entire mineral wealth produced over this period that has seen such tremendous industrial development in our country. As a matter of fact, our present industrial civilization has been made possible through the use of the

mineral fuels for heat, power, metallurgical and chemical purposes. The tonnage of coal hauled by our railroads is greater than the freight tonnage of all manufactured goods.

If this record of an industry could be translated into a more graphic picture, which would show the jobs created, the homes heated, the ores reduced to metal,



1.—COMPARATIVE VALUES OF WORLD PRODUCTION OF METALS, FUELS AND OTHER NONMETALS (1929).
(U. S. Bureau of Mines *I.C.* 6643, June 1932.)

The transport of oil and oil products as well as of natural gas from the producing to the consuming areas is a major industry with hundreds of thousands of miles of pipe lines.

In 1939, there were 422,000 productive workers employed at bituminous coal mines, 93,138 at the anthracite mines, 117,570 oil-field workers (1938), 8,090 at gas wells (1938). At least 4,000,000 people consisting of productive workers and their families are dependent directly on the mineral-fuel production for their livelihood. When we consider the production, sale and transportation of these raw materials, we see some ten million or more people dependent directly or indirectly on this industry. Certain railroads receive a major fraction of their income from the transportation of the solid fuels. And huge sums of money are paid into the public coffers in the form of taxes directly by the industry and indirectly through the medium of taxes on people employed by the industry or, in connection with petroleum products, through taxes on consumers.

the engines moved, the cities and villages lighted, the drugs, dyes, explosives, plastics and other chemicals manufactured, the freight and passengers transported, we would have some indication of the economic significance of the mineral fuels in modern civilization.

Need for Technologists

The need for technologists specially trained for the petroleum industry has been recognized for at least 15 years and petroleum (and in some instances natural gas) engineering curricula are now well established in a number of universities. The petroleum industry early made use of technology and constant research and application of improved methods have resulted in steady improvement in quality of wares and in decreased costs as well. Petroleum is in some respects an ideal raw material, for it is possible, within certain natural limits, to change the yields of the various products by varying the processes of treatment—temperature, for example. The petroleum industry has been flexible and has always been able to meet the

demands made upon it by the ultimate consumer. Thus when the demand was for kerosene, the industry produced principally kerosene. When the lighting market was lost to gas and electricity and the demand for motor fuel and lubricants stepped up at an increasing pace, the industry found ways of increasing the yields as well as improving the quality of these products.

The coal-mining industry (referring now to all ranks of coal), on the other hand, has no similar history of technologic advance, except in so far as mining is concerned. There, indeed, it has made phenomenal progress in mechanization and other improved mining methods. One of the reasons for this failure of the industry to plow part of its profits back into the business in the form of research is undoubtedly the scarcity of technical men in responsible positions of management. I know of only one coal company that has as its president a man who is a scientist in his own right. This is one of the obstacles to a full appreciation of the value of scientific work and to the selection of properly trained technologists to conduct the work. Very few coal companies, including those maintaining laboratories, have technologists qualified to originate, pass upon or carry out research programs. Small wonder then that accumulation of data is mistaken for research and that the management of companies is often poorly advised with reference to matters of research policy.

However, it must not be assumed that no technologic advances have been made in the utilization of fuels, for great improvements have been recorded for all types of consuming equipment. These have been most marked in central electric power stations where the coal required per kilowatt-hour has decreased from over 6 lb. in 1902 to 1.37 in 1940. Good economies have been effected in the operation of steam locomotives (180 lb. of coal per 1000 gross ton-mile in 1917 to 112 lb. in 1940), and iron blast furnaces (3400 lb. coking coal per gross ton of pig iron in 1915 to 2846 lb. in 1940). In fact, most of the large industrial consumers of coal have increased the efficiency of utilization, with a subsequent decrease in their tonnage requirement. These advances have been made by the consumers themselves, largely without any aid on the part of the coal industry. Moreover, little attention has been paid by the producer or any one else to the needs or desires of the small consumer—nor has any agency given attention to coal as a foremost raw material for manufacture of organic chemicals. On the other hand, the petroleum industry has come to regard crude oil as just that, and now makes chemical products that are in competition with products from coal; toluene, for example.

The technology of all fuels has grown enormously during the past several decades. At the same time every phase of recovery, preparation, transportation, marketing and utilization has become more technical and complex. Hence, the need for well-trained men of many types is increasing. There is room for a few well-placed curricula in Fuel Technology, so that men trained in fundamentals and speaking in the terms of the fuel industry will be available in the future.

Historical Development of the Curriculum

Such courses have long been an important part of European university instruction. The curriculum originated, naturally enough, out of demands of the gas industry, which, owing to the nature of the processes involved, required the services of men with a thorough understanding of the scientific principles underlying the processes of manufacture. Thus, in Germany, at the instance of the gas industry, the now famous "Gazinstitut" in Karlsruhe was founded in 1904. Three years later a department for teaching and research in

Fuels and Metallurgy, the first of its kind in the British universities, was established at the University of Leeds. This department had the support of the West Yorkshire coal owners as well as of the Institution of Gas Engineers. The scope of the curriculum is indicated by the present title of the department—Coal, Gas and Fuel Industries with Metallurgy. Somewhat later a Department of Fuel Technology was established at the University of Sheffield.

In the United States, on the other hand, although courses relating to fuels have been offered in a number of universities and colleges for many years, these have usually been given in other departments, as for example chemistry, mechanical engineering or metallurgy. Until very recently, there has been no general recognition of the fact that there exists a distinct need for a curriculum in Fuel Technology in a few institutions of higher learning strategically located with respect to the major fuel-producing and fuel-consuming areas. And yet the American gas industry recognized this need at least 17 years ago when a curriculum in gas engineering was established at The Johns Hopkins University. The support of the gas industry was later withdrawn but the curriculum is still being maintained.

In The Pennsylvania State College, coal washing was an important subject in the mining engineering curriculum as long ago as 1893. This was extended in 1894–1895 to include the erection of "a gas producer, regenerator and furnace for investigation into the economical use of fuels" (quoted from the general catalogue of that year). By 1906, the importance of instruction in the technology of fuels was generally recognized by the College. At that time lecture and practicum courses in coal washing and washing machinery, the preparation of coal for the market, and an elective 10-weeks summer practical course in coal washing were offered under the heading of ore dressing and coal washing. A course in mining was concerned with mechanical treatment and preparation of anthracite and bituminous coal; one in mining geology included the origin, occurrence, distribution, and properties of coal; and courses in metallurgy involved the study of gas producers as well as fuels and fuel testing.

Before the academic year 1908–1909, the courses offered by the School evidently were reorganized. During that year the Department of Mining offered a course titled "Mechanical Preparation of Coal and Dressing of Ores," which suggests that a consolidation of courses must have been effected. A practicum course pertaining to this subject was also offered. Under Metallurgy there was offered a course in Principles of Metallurgy, Fuel Testing and Calorimetry. The following year, however, a separate course in fuels and fuel testing again appeared in the College catalogue.

From 1910 to the college year 1918–1919, the courses relating to fuel technology remained substantially unchanged. During the latter year several changes in course title and designation were made and specific courses on fuel testing and calorimetry and on coking were added. During the decade 1920 to 1930, there was growing recognition on the part of the faculty of the importance of advanced instruction and research as well as undergraduate instruction in fuel technology. This is evidenced by the introduction of graduate courses in coal carbonization in 1921 and in utilization of fuels in 1922, by the Department of Metallurgy, and by the publication of several bulletins.

These developments resulted in 1930–1931 in an option in Fuel Technology as part of the metallurgy curriculum. In this option courses were offered to undergraduate students in fuel testing and calorimetry, coking, classification of coals, liquid and gaseous fuels, carbonization and processing

of coals, combustion and utilization of solid, liquid, and gaseous fuels. A graduate course was offered in metallurgical utilization of fuels.

Fuel technology had now come of age. Its particular importance as an integral part of mineral industries education in Pennsylvania was recognized by the Board of Trustees of the College and in 1932 a separate curriculum was established for the first time. With minor changes, the courses offered were the same as those offered under the metallurgy option. The Department of Mining offered a lecture course in coal preparation, a practicum course on the same subject, and another course in coal cleaning. The courses offered by the new Department of Fuel Technology were given in the junior and senior years but before that the student had acquired a thorough background in chemistry, mathematics, physics, engineering, geology, and certain cultural subjects.

Although curricula in fuel technology have been offered in English and German universities for many years, the curriculum at The Pennsylvania State College was and is the only general program of this type offered in this country. It was created because of the importance of the fuel-producing and fuel-consuming industries in the state of Pennsylvania and the growing need for men trained in the technology of fuels. At the very outset graduate instruction and training in research methods were offered.

At present, by a process of careful selection and revision of courses in accordance with experience and the needs of the fuels industry, the curriculum in fuel technology has been well established. It has been accredited and has received the commendation of men in the fuels industry who are in a position to appreciate the necessity of a source of young men possessing a knowledge of the origin, constitution, preparation, processing, and utilization of fuels—especially coal.

Scope and Content of the Fuel Technology Curriculum

The scope and content of the fuel technology curriculum is a problem of the first magnitude, and one for which there is no single solution. It involves consideration of needs and trends of the industry, some of which have already been cited. The fuels industry, prior to the present emergency, was faced with a highly competitive situation within itself, which threatened it with disaster. Instead of suffering from the oft-predicted shortage of coal, petroleum and natural gas, the country had a capacity for production greatly in excess of its needs. Coal began to meet greatly increased competition from petroleum, natural gas and water power. The total energy demand had increased steadily during the past several decades, but the proportionate share of this demand met by coal decreased markedly. During this time the petroleum and natural gas industry expanded enormously and in 1929 supplied about 19 per cent of the world's total demand for energy. In spite of this, however, the oil and gas industries during the past decade were in much the same position as was the coal industry. Overproduction, falling prices, financial losses and unemployment characterized the entire fuel industry. Conditions in 1941 were different, to be sure, but there is every reason for expecting after the present war a recurrence of the situation extant in the 1930's. These are economic factors that must be given very serious consideration in future education programs for the training of scientists, engineers and technologists.

There is still another factor, which, heretofore, has been largely if not entirely neglected; that is, the factor of social utility. In the history of science and invention, the social consequences have been given little consideration. Now, however, mastery over material forces has far outdistanced the old moral and legal

controls and the future welfare of society demands a type of scientific leadership that is fully awake to the importance of some sort of moral or social control: whether this will or will not lead to further public regulation than exists at present is an open question. There seems no doubt, however, that the engineer of the future will have to consider factors of economic effects and social expediency to a much greater extent than the engineer of the past. Other factors that must be considered are ones concerned with changing consumer habits, as exemplified by the desire for fully automatic heat.

I have spoken at some length concerning matters which at first glance may appear to be irrelevant. Those of you who have read "The Education of Henry Adams" may remember his complaint in after life that his years at Harvard College did not prepare him for his life work. Although there may be differences of opinion concerning the objectives and methods of the college and the purpose and advantages of a college training, there can be no doubt that the college training to be effective must fit one not for the present environment but for one that is yet to come. In studying curriculum problems, therefore, it is of greatest importance to project oneself into the future and attempt to anticipate, however inadequately, some of the demands that may be made upon graduates in the future. These matters are of general application to all applied sciences. The fundamental scientist need not be concerned therewith except insofar as support for his researches depends upon any changes that may take place in our national habits of underwiting such work. Since he is engaged in increasing the world's supply of knowledge, he may rest assured that sooner or later his work will reach some utilitarian application.

How, then, shall we train our future fuel technologists so that they are equipped to meet the demands that the future will make upon them? That is indeed a difficult question to answer, since no one, especially during these times, can foretell what the future will be like. Furthermore, other limitations are placed on our choice of subjects to be included. One of these is the duration of the curriculum. Here we are limited by the fact that the majority of the engineering schools and colleges still adhere to the four-year curricula and industry seems to feel that the engineering student should secure his essential academic preparation in a terminal four-year program. Yet, it is increasingly apparent that four years of collegiate preparation are not enough to accomplish more than a superficial outline of the professional field. Five years would make possible a much more thorough training. The gas industry has recognized the shortcomings of a four-year program and has established an Institute of Gas Technology—in reality a graduate school for the industry—of which the objectives are: training of men for the industry, fundamental and applied research, collection and dissemination of scientific information and encouragement of further research by others in the business.

Regardless, however, of the length of time alloted to the curriculum, the difficulties of predicting the future make it imperative that any curriculum devote a large part of its program to the basic subjects that time has demonstrated to have lasting value; that is, mathematics, physics, chemistry, earth sciences and English. To these, then, will be added the general engineering subjects that are a part of every engineer's training, economics and the professional courses in fuel technology, which will acquaint the student with the problems and terminology of the fuels industries and give him some training in design. It would also be desirable for the undergraduate program to contain general courses in such subjects as public speaking, in the humanities, in industrial

management, and similar subjects, but we must keep in mind the four-year limitations.

Opinion will vary, of course, among educators and executives as to the amount of time to be given to the various subjects, and even to the subjects to be included. The writer has given the subject considerable thought over the past decade and has discussed it with many engineers, educators and executives. In general, the fuel technologist must possess some knowledge of the occurrence and methods of winning the various solid, liquid and gaseous fuels. In detail, however, this is really the realm of the mining and petroleum and natural-gas engineer. Similarly he must know something of the ordinary methods of utilization of fuels in the production of power, although that is the domain of the power-plant engineer.

The field of knowledge in between these two, however, must be familiar in full detail to the fuel technologist. To begin with, he must be thoroughly grounded in the proper-

TABLE 1.—*Curriculum in Fuel Technology at The Pennsylvania State College*

Freshman Year

First Semester	*Credits*
Chem. 1, Inorganic Chemistry	5
Dr. 1, Engineering Drawing	2
Engl. Comp. 1, Composition and Rhetoric	3
Geol. 31, Physical Geology	3
Math. 4, Plane Trigonometry and Algebra	4
M.I. 1, Mineral Industries Lecture	1
Physical Education 1	1
R.O.T.C. 1	1½
	20½

Second Semester	*Credits*
Chem. 2, Inorganic Chemistry and Qualitative Analysis	5
Engl. Comp. 5, Exposition	3
Geog. 9, Geography of Mineral Resources, or Dr. 2, Descriptive Geometry	2
Geol. 32, Historical Geology	3
Math. 7, Analytic Geometry	4
Physical Education 2	1
R.O.T.C. 2	1½
	19½

Sophomore Year

First Semester	*Credits*
Math. 29, Differential Calculus	3
Min. 31, Elementary Mineralogy	3
Phys. 231, General Physics	3
Phys. 232, Physical Measurements	2
C.E. 111, Plane Surveying or approved electives	4
Physical Education 3	1
R.O.T.C. 3	1½
An approved elective	3
	20½

Second Semester	*Credits*
Chem. 20, Quantitative Analysis	5
Math. 30, Integral Calculus	3
Fuel T. 1, Fuel Testing and Calorimetry	2
Phys. 281, General Physics	3
Phys. 282, Physical Measurements	2
Physical Education 4	1
R.O.T.C. 4	1½
An approved elective	3
	20½

TABLE 1.—(*Continued*)

Junior Year

First Semester	*Credits*
Chem. 32, Carbon Compounds	2
Chem. 40, Introductory Physical Chemistry	3
Chem. 42, Experimental Physical Chemistry	1
E.E. 8, Dynamo Machinery	2
El. Lab. 8, Electrical Engineering Laboratory	2
Fuel Tech. 2, Advanced Fuel Testing	3
Fuel Tech. 3, Solid and Liquid Fuels	2
Mchs. 7, Mechanics	3
	18

Second Semester	*Credits*
Chem. 33, Carbon Compounds	3
Chem. 41, Introductory Physical Chemistry	2
E.E. 9, Industrial Electrical Applications	2
Engl. Comp. 23, Report Writing	2
Fuel Tech. 6, Distillation of Coals	2
Fuel Tech. 80, Junior Field Trip	1
Mchs. 8, Mechanics	3
Mng. 1, Elements of Mining	3
	18

Senior Year

First Semester	*Credits*
Fuel Tech. 4, Thermal Reactions of Fuels	3
Fuel Tech. 99, Seminar	1
Fuel Tech. 401, Gas Manufacture	3
Fuel Tech. 403, Energetics of Fuel Technology	3
Mng. 85, Mineral Preparation	2
Mng. 86, Mineral Preparation Laboratory	1
M.E. 101, Elements of Power Engineering	2
M.E. 103, Power Laboratory	1
Met. 59, Engineering Metallurgy	2
	18

Second Semester	*Credits*
Chem. Eng. 3, Chemical Engineering	4
Fuel Tech. 5, Fuel T. Design	3
Fuel T. 81, Senior Field Trip	1
Fuel T. 100, Thesis	3
Fuel T. 402, Processing of Coals	3
Mchs. 3, Engineering of Materials	1
Mchs. 5, Testing Materials	1
Mng. 492, Advanced Mineral Preparation	3
	19

ties of the different fuels and the relationship between them. Then, with respect to coal, which is still the most important of the fuels, preparation and processing have become as important as mining and utilization. The fuel technologist must have a thorough understanding of the underlying principles of coal preparation and washing. This, in my opinion, involves some knowledge of the fundamental nature of coal, including the types, composition and physical properties of the mineral matter associated with the coal substance and the effect of the mineral matter on coal utilization.

Similarly, coking involves not only a knowledge of the different carbonization processes but, what is much more important, fundamental information concerning the physics and chemistry of the distillation of coal, the important properties of coal that effect in any way the coking process, as well as the recovery, processing and treatment of the by-products recovered in carbonization.

Also, the subject of combustion involves much more than boiler tests and standard combustion equipment. The fuel technologist must know the mechanism of combustion of solid, liquid and gaseous fuels and its relation to equipment design, besides the properties of the fuel that influence its selection for various types of utilization.

For purposes of discussion the curriculum now in effect at The Pennsylvania State College is presented in Table 1. Admittedly, it is not perfect! Curricula, after all, are man-made and subject to local conditions as well as to the special interests and background of the individuals that originate them.

For convenience, Table 2 is given, summing up the curriculum shown in Table 1. The time allotment, exclusive of military and physical training, contemplates eight semesters with 18 credit-hours of time available in each.

Fourteen credit-hours of advanced algebra, plane trigonometry, analytic geometry,

TABLE 2.—*Undergraduate Program in Fuel Technology*

	Semester Credit, Hours
Mathematics	14
Physics	10
Chemistry and Chemical Engineering	30
Earth Sciences	12
English	8
Electives, Foreign Languages, Humanities, Administration, Management	10
Drawing, Mechanics and Strength of Materials, and General Engineering	30
Fuel Technology	30
Physical and Military Training	10
	154

and the integral and differential calculus should be sufficient mathematics for all ordinary problems that the fuel technologist will meet in his professional career. As preparation for graduate work it would be desirable to include three credit-hours of differential equations, if the time were available.

Ten credit-hours of physics and 30 of chemistry and chemical engineering cover the various fields of physics needed for engineering work as well as general chemistry, qualitative and quantitative analysis, physical chemistry, introductory organic chemistry and the basic principles underlying unit operations in industrial chemical processes. This is very much more chemistry than the average engineer is exposed to—a weakness in much of our engineering training—but most of the processes through which fuels go in the course of their preparation and ultimate utilization are chemical processes.

Twelve credits of earth sciences include physical and historical geology, geography of mineral resources and elementary mineralogy. Some engineers and fuel technologists may question so large an allotment of time to these subjects but geology is essentially a cultural subject. We live on the earth and every one should have some knowledge of it. Furthermore, the mineral fuels are obtained from the earth and the fuel technologist may run up against problems related to their occurrence. It is doubtful whether an introduc-

tion to these subjects can be obtained in less time.

The allowance of 8 credit-hours for English and 10 credit-hours electives in foreign languages, humanities, administration and management is admittedly less than might be desired. It does, however, permit an introductory course in economics, some work in English composition and report writing and some opportunity for election of courses by the student.

The 30 credit-hours of drawing, mechanics and general engineering subjects are adequate for groundwork in mechanics and strength of materials, as well as an introduction to mechanical, electrical and mining engineering, and metallurgy and mineral preparation. It is conceivable that some work in ceramics (on refractories) and in petroleum and natural-gas engineering would also be desirable, but once more we are faced with the limitation impressed upon us by the restriction of a four-year program.

The 30 credit-hours of fuel technology are designed to give a thorough grounding in fuel sampling and analysis, the reactions that fuels go through on heat-treatment, the manufacture of gas from solid and liquid fuels, the miscellaneous processing of coals such as briquetting or hydrogenation, application of thermodynamic principles to fuels problems, and fuel technology design. Opinions will vary as to the time to be allotted to the purely professional subjects as well as to their nature. Thus the curriculum in gas engineering at The Johns Hopkins University gives particular attention to that specific branch of fuel technology. The tendency there is to give more chemical engineering subjects. On the other hand, at the University of Iowa there is an option in Fuel Technology in the Department of Mechanical Engineering. The fuel technology courses are decidedly meager and there is a woeful lack of training in chemistry. The curriculum is in reality one in mechanical engineering with the election of a few specific courses in fuel technology during the junior and senior years. These include fuel production and utilization, theory of combustion, heat transfer and design of industrial furnaces, and preparation of fuels and disposal of wasted products of combustion. While such training may be satisfactory for a mechanical engineer who is going into power-plant operation or furnace design, it will hardly suffice for an all-round fuel technologist. The course leans entirely too heavily on mechanical engineering, and illustrates one of the difficulties arising from considering fuel technology as a stepchild of some established curriculum, whether metallurgy, chemical or mechanical engineering. I have heard one engineer and educator question the need for a specialized curriculum, although he did not question the opportunities in the fuels field. This man believes that:

> Any engineering student gets all the basic science that is necessary as a foundation. Either a chemical engineer or a mining engineer has most or all of the supplementary chemistry that is needed but could probably profit by additional courses in heat power engineering and fuels. Mechanical engineers have abundant training in the heat power field but usually are somewhat deficient in quantitative and fuel chemistry, and have only a sketchy idea of fuels. For a young man in any one of these engineering curricula who looks forward to fuel engineering as a career, the proper selection of a few hours of elective work in the fields in which his curriculum is short would be sufficiently well prepared.[1]

I consider this attitude to be exceedingly short-sighted; such a policy can only produce inadequately trained men, whereas the fuels industry needs technologists with first-class training and with a feeling for the dignity of the profession of fuel technology.

Rose has another point of view, to which I subscribe. To quote from his remarks

[1] Transactions Third Annual Anthracite Conference of Lehigh University, 1940, p. 42.

before the Gas and Fuel Division of the American Chemical Society in April 1940:

Some educators and executives favor a training in fundamentals, with no technologic courses during undergraduate years. Others favor offering, to a limited number of interested students, fundamental training plus courses intended to prepare them for special work such as fuel technology. In the writer's own experience, which has included all three major types of solid fuels, both methods of training have produced valuable men.

Large engineering or research organizations, which are well staffed and progressive, can take a young engineer or research man who has had no previous contact with their field of work, and train him to become a valuable member of their organization. Executives who favor this procedure sometimes forget that for every organization which is qualified to do this, there may be a dozen or more small producers, consumers, retail dealers, etc., who need a single technical man badly. They are in no position to give him engineering training; instead they must look to him for information. If they are not in a position to hire an experienced man away from a competitor, they will prefer a young technical man who has already shown interest in their industry, and who has invested some of his own time preparing for it. They will want a man who already has some familiarity with their problems, if only through textbooks. He will at least have a few reference books of his own, and will know where to go for specialized technical help when he needs it.

This is one reason for the existence of a somewhat specialized curriculum like Fuel Technology (which is the science of preparing and utilizing fuels). Another purpose is to provide elective or post-graduate courses for students majoring in other lines of engineering or science, who recognize the basic importance of fuels.

It seems evident that if fuel technology is to develop and fulfill its mission it must do so through the efforts of fuel technologists. It must be recognized as a technology in its own right and it must be favorably spoken of and placed before the industry and the secondary schools.

The writer has yet to touch upon the development of a sense of social responsibility as part of the training of engineers. Table 1 reveals no courses in this subject—nor does it seem possible that there can be such specific courses. Each faculty member must strive in each course to instill in his students a sense of social responsibility. The titles, of courses, are, after all, unimportant. The quality of higher education depends upon faculty, equipment and students. The faculty must be alert, familiar with industrial processes in its field, alive to new developments, active in research and above all sympathetic and friendly with students. If, in addition, the faculty members themselves have a sense of social responsibility, they can do much in their respective courses to awaken such a sense in the students. This is particularly true if the classes are not too large.

Who Employs Fuel Technologists?

At this point it seems only reasonable to inquire who employs fuel technologists. Certainly the solid fuels industry will require an increasing number of fuel technologists. Such men are needed by the producing companies, sales agencies, and equipment manufacturers who design and manufacture equipment for preparation, treatment and utilization of coal. The railroads use them to promote the sale of coal originating on their lines and to aid in the selection of coals and the design of combustion equipment. By-product coke plants and gas companies, power plants, steel companies and other metallurgical plants, also have extensive use for fuel technologists.

The sale of fuel by retailers and by wholesalers in the future will be guided by technologists. The installation of oil burners and coal stokers, the development of air conditioning as well as other modern advances in home heating, hot-water supply, etc., will make it necessary for progressive fuel dealers to sell service rather

than merely to take orders. There will be excellent opportunities in this field for the young technical man who can make simple layouts and cost estimates, advise builders and architects and develop the ability to sell service as well as competitive engineering products.

A large number of fuel technologists, including some of the best known, are connected with federal and state agencies, or with colleges and universities, or are connected with the research and development programs of various trade and marketing associations.

That there is a demand for the graduates is evident, for at no time during the past 9 years have we had enough at The Pennsylvania State College to fill all the requests from companies seeking to employ them.

Graduate Study and Research

Graduate study and research are also important in modern education. To serve society adequately in the solution of the social and economic problems facing the mineral industries, our colleges must educate leaders by graduate as well as undergraduate instruction and must supply leadership in the solution of technical problems by research.

During the early part of the present century the need for research was not felt by the fuels industry, particularly the coal industry. Enormous deposits of high-grade coal were available and the best seams and those most easily worked were being mined with profit. Coal was king and oil and natural gas were just beginning to offer serious competition. Today the picture is changed and the dominant need of the industry is for research.

The two chief objectives in graduate study are to strengthen the student's grasp and to broaden his knowledge of his general subject; and to develop his ability to apply existing knowledge in new directions. The primary approach in the first instance is made through supervised study; in the second, through original research. To accomplish these objectives, the student is advised to select courses in those fields of science and technology in which he is deficient. Specific courses in fuel technology are limited in number, but the student is required to devote a considerable share of his time to research on some problem in this field. It is believed that "learning by doing" offers the best method for research training. At the same time the student is encouraged through seminars and individual conferences to read widely in the current literature in fuel technology.

To sum up, the writer has shown that the body of knowledge that comprises the technology of fuels has become so extensive that its consideration as a separate branch is warranted; conditions in the fuels industries have changed during the past several decades, so that men specifically trained in fuel technology are needed; there is a need for a curriculum in fuel technology in a few colleges and universities strategically located with reference to fuel resources; the objectives, scope and content of such a curriculum are discussed; opportunities for employment are indicated.

DISCUSSION

E. G. Bailey,* New York, N. Y.—There is much in Dr. Gauger's paper with which I fully agree, but I would like to take the liberty of discussing some points with a view to clarification and others from the viewpoint of constructive criticism.

The metallurgist was the first user of coal requiring knowledge of its chemistry. The usual analysis is largely based on the needs of the metallurgist, and such analyses, tabulated in many forms from various sources, comprise the most complete chapter in coal technology. In fact, many people have a misconception that chemical analyses in themselves are the end instead of only the first stepping stone toward other knowledge so urgently needed by many users. The gas and coke industries have done

* Vice President, The Babcock & Wilcox Co.

well in carrying the chemistry and properties of coal further for their specific requirements.

Due largely to the mechanical engineer and the use of coal for steam generation, the chemical laboratory work has been extended to include calorific value, then the fusing temperature of coal ash, and later grindability, friability, agglutinating properties. We still need more knowledge of clinkers and slag behavior under different conditions and atmospheres in which coal is burned.

We know much about the making of gas and coke and the complete combustion of the fuel elements, but we must rapidly learn much more about the combustion and chemical reactions of the impurities, such as the FeS_2, FeS, Fe, FeO, Fe_2O_3 series, as well as the alkalies and their reactions in oxidizing and reducing atmospheres.

We must know more about ignitibility, reactivity of coal and coke in pulverized form. We know much less than we should about accurate furnace temperatures, gas stratification, and the actual path of gases in furnaces and its effect upon results.

Dr. Gauger sees great advance in the use of coal by central electric companies made by themselves largely without any aid on the part of the coal industry. He seemingly overlooked the part played by the manufacturers of coal-burning and steam-generating equipment. The early stoker development was largely by manufacturers of stokers without much aid from the boiler manufacturers, and little from the users except as the latter bought and tried out new developments in an effort to use cheaper coal, obtain better efficiency, and to save labor. Modern development of water-cooled furnaces for pulverized coal, and the multi-fuel burning equipment for coal, oil, and gas, can only be brought about through a unifying of design by the manufacturer of fuel-burning and steam-generating equipment. Central stations, with their rapid expansion, have been the best purchasers of this rapidly developing line of equipment, because it paid them well to follow such advances. Smaller power users have also followed this progressive trend, but more slowly.

This development has not been done by mining engineers, chemical engineers, mechanical engineers, nor even by "fuel technologists," as Dr. Gauger trains them, but by groups of men and individuals who have combined knowledge of fuels, combustion, ash and slag behavior, thermodynamics, heat transfer, boiler circulation, water treatment, steam purification, and the flow of fluids under a large variety of conditions. A modern steam-generating unit cannot be an aggregation of parts of an equipment of which each is designed by a specialist; it must be an integrated whole wherein all of the many requirements are met in a satisfactory and economic manner.

In the conventional steam locomotive, for instance, the coal-burning and steam-generating problems are so closely linked together that one cannot be changed without seriously affecting another. As a result of the lack of men trained well enough in modern developments of both lines jointly, the iron horse has remained a most inefficient coal-burning unit, requiring the most elaborately prepared and expensive fuel for its highest possible capacity, and at the same time limiting the power and efficiency of the boiler because of the low steam pressure, low superheat, and high carry over of moisture and solids from the boiler to the engine.

Who Employs (or Should Employ) Fuel Technologists?

A. Coal Producers

1. There is no disagreement on this question, but it is my observation that in this industry the fuel engineers now employed are better trained in production, cleaning, and simple chemistry than in the use of coal.

2. Those contacting or servicing users should be well versed in the broad field of the use of coal in the plants of their customers. The present college training facilities are inadequate for the best service to this industry. Men for this class should have postgraduate courses in the way of jobs with equipment manufacturers or large consumers.

B. Equipment Manufacturers

1. Mining, loading, preparation, crushing, screening, cleaning, treating, etc.

2. Processing, gas, coke, etc.

3. Metallurgical and ceramic.

The present training in many schools is adequate to start satisfactorily in these industries, but a still better curriculum is desirable.

4. Coal burning for steam generation and direct heat absorbers, viz: pulverizers, stokers, boilers, superheaters.

Present training inadequate, and Dr. Gauger's curriculum might well be modified to include less geology, surveying, mining, and even processing, and include more thermodynamics and power engineering.

5. Domestic and small direct-heating units. Dr. Gauger's curriculum probably adequate, with some reduction mentioned for *B*-4, and more inspiration to invent and develop and market a real worth-while domestic coal-burning unit.

C. Consumers

1. Processing, gas, coke, etc.

2. Metallurgical and ceramic. Requirements similar to *B*-1, 2, 3.

3. Steam-power plants: (*a*) central stations; (*b*) industrial plants. Requirements similar to *B*-4.

D. Railroads

1. Extension work as partners with coal producers. Requires similar training and experience to *A*-2.

2. Locomotive, design and operation. Training should be more like that desired for *B*-4.

E. Technical—Laboratories

1. Commercial laboratories.

2. Consulting engineers.

3. Teaching.

Dr. Gauger's curriculum is probably adequate for beginners in *E*-1, but of course for *E*-2 and *E*-3 a wider *experience* is needed, and that carries us beyond curriculum for bachelor's or even doctor's degree.

The classifications and requirements given above may be reclassified regardless of employment by producer, equipment manufacturer, user, etc., into the three major uses of coal, viz:

Uses	Approximate Percentage of Total Coal Production
1. Process (steel, coke, metallurgy, etc.)	23
2. Steam power and larger heating boilers:	
Electric utilities	12
Railroads	20
Manufacturing industries	25
3. Domestic and retail	20

In my opinion Dr. Gauger's curriculum is better for training men for work in the process uses than for the steam-power group. The importance of the latter fully justifies a curriculum previously stated in discussing *B*-4.

The definite importance of thermodynamics and power engineering for certain work, beyond that now prescribed in any known chemical engineering, fuel engineering, or fuel technology course, has made it seem necessary for us to select mechanical engineers, preferably with the maximum of chemistry as electives, and teach them more of the chemistry and metallurgy and ceramics as needed and from actual experience, than to take on a chemical engineer or fuel technologist and try to ground him in thermodynamics and power engineering.

Most engineering courses need more than is now prescribed in physics, economics, English (spoken, written, and vocabulary), as well as management and administration.

In my opinion, Dr. Gauger's history of the development of fuel engineering or technology is inadequate, if not actually misleading. It is a subject of sufficient importance for a committee to take up the task of compiling a fairly complete and accurate chronology of development in fuel engineering curricula.

Such a history would indicate that while certain progress in Europe preceded that in this country the greater progress has been made here. Great credit goes to the geologists, viz: M. R. Campbell, J. A. Holmes, I. C. White; to the U. S. Geological Survey—1904—later U. S. Bureau of Mines; also to Professors Parr and Breckenridge, of the University of Illinois, and Professors N. W. Lord and Hitchcock, of The Ohio State University.

Undoubtedly more fuel chemists and engineers gained their early inspiration and training from the men mentioned above than from all others combined.

A. W. Gauger (author's reply).—Mr. Bailey's stimulating and provocative remarks seem to me to be a potent argument in favor of a specialized curriculum in fuel technology. His admission that the advances in the use of coal that have been made by equipment manufacturers were accomplished "by groups of men and individuals who have combined knowledge of fuels, combustion, ash and slag behavior, thermodynamics, heat transfer, boiler circulation, water treatment, steam purification, and the flow of fluids under a variety of conditions"

certainly bears out my fundamental thesis, which is that there is an increasing need for specialized training in the technology of fuels.

Mr. Bailey's suggestions as to content of the curriculum are both suggestive and interesting. They reflect the point of view of his very considerable experience and must be given serious consideration. However, to eliminate such a cultural subject as geology (the writer believes that since man lives on the earth he should know something about it) does not seem wise. Surveying can well be eliminated and is optional in our curriculum. Some knowledge of mining and preparation is desirable even for combustion engineers, because it indicates limitations beyond which the "manufacturers of coal" cannot go without adding to the cost of the fuel. Even design of fuel-burning equipment must take cognizance of the fuels that are available.

With reference to Mr. Bailey's final remarks on the history of the development of fuel technology, may I point out that the section referred to is entitled "Historical Development of the Curriculum." We have had excellent fuel technologists and Mr. Bailey himself is one of the pioneers. However, these men have been strong personalities with a consuming interest in fuels. They accomplished their results despite the prevailing attitude toward the science of fuels. And following the passing of men like Parr and Lord there was no continuity such as would have been provided had they been heads of established departments of fuel technology or fuel engineering.

Correlation of the Bureau of Mines-American Gas Association Carbonization Assay Tests with Coal Analyses

By H. H. Lowry,* Member A.I.M.E., H. G. Landau,* and Leah L. Naugle*

(New York Meeting, February 1941)

Evidence has been accumulating in recent years, in part from the work of the Coal Research Laboratory, that coals belong to a family of natural polymers and that even in complex reactions the differences between coals are quantitative and not qualitative. This point of view leads to the expectation that if a series of coals were subjected to a closely controlled set of experimental conditions the results obtained should be related to the chemical analyses of the individual coals. The data accumulated over the past 10 years by the Bureau of Mines in its "Survey of Coke-, Gas-, and By-Product-Making Properties of American Coals," in what has come to be recognized as the Bureau of Mines-American Gas Association (BM-AGA) carbonization assay test,[1-12] offer an unusual opportunity to check this conclusion. The present paper reports an analysis of the data on the first 90 coals and coal blends—including coals numbered 1 to 55B, inclusive—which have been carbonized according to a standard procedure at intervals of 100°C. from 500° to 1100° in either or both 13 or 18-in. cylindrical retorts.[13]

In Table 1 are given the name of the seam from which the sample was obtained, the analytical data used, and the reference to the original publication of the Bureau of Mines. The coals range in fixed carbon from 46.2 to 79.0 per cent, in volatile matter from 15.4 to 41.3 per cent, in ash from 2.1 to 15.9 per cent, in moisture from 0.8 to 19.7 per cent, and in total sulphur from 0.40 to 4.44 per cent; therefore the entire gamut of bituminous coals is represented. The petrographic nature of the individual coals is not considered in the following analysis, although both "normal" bright coals and splint coals are included in the samples studied. For information on the petrography of the coals and for further information characterizing the coals the original papers should be consulted.

Although the coals that are included extend over the whole range of bituminous coals, they do not cover this range entirely adequately. In particular, only 13 of the 90 coals have moisture contents greater than 3 per cent. This concentration at low moisture values means that the effect of moisture, which is an important factor, cannot be determined with as much certainty as would be desirable.

All the coals were not tested at every temperature and in both retort sizes. Only the first 30 coals were tested at 1100°, and only about 50 at the other temperatures, except at 900°, where nearly all of them were included. In the accompanying tables the actual number of observations entering into each correlation is given.

In this study equations are given for calculating analyses, yields, and properties of coke and by-products from the coal analyses. To decide whether these equations are satisfactory it is necessary to know the accuracy with which these quantities were measured in the assay tests.

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1 References are at the end of the paper.

TABLE 1.—*Analyses of Coals Used in Carbonization Assays*

Bed	Coal No.	Moisture	Volatile Matter	Fixed Carbon	Ash	Nitrogen	Total Sulphur	B.t.u. as Received	Ref. No.
Pittsburgh	1	1.6	34.2	56.7	7.5	1.6	1.1	13,810	1
Pittsburgh (washed)	1W	2.4	34.5	57.2	5.9	1.6	1.0	13,910	1
Elkhorn	2	2.2	36.6	59.0	2.2	1.5	0.6	14,410	1
Taggart	3	1.7	36.1	59.5	2.7	1.5	0.6	14,770	1
Taggart	3A	1.2	36.6	59.5	2.7	1.4	0.6	14,800	1
Davis	4	1.4	22.1	66.7	9.8	1.5	1.45	13,820	1
60 % No. 4, 40 % No. 1W	5	1.5	27.7	63.3	7.5			14,070	1
20 % No. 4, 80 % No. 1W	6	1.3	33.3	60.0	5.4	1.6	0.9	14,220	1
Mary Lee	7	1.2	27.9	55.0	15.9	1.5	0.74	12,600	1
Mary Lee (washed)	8	4.2	27.6	59.9	8.3	1.6	0.78	13,550	1
Pittsburgh	9	1.9	33.6	57.0	7.5	1.6	0.97	13,910	1
Illinois No. 6	10	7.9	32.1	47.7	12.3	1.4	0.82	11,540	1
B (Michel)	11	1.4	27.0	65.4	6.2	1.3	0.57	14,240	1
Pittsburgh	12	1.0	33.6	56.9	8.5	1.6	1.25	13,820	1
Black Creek	13	3.1	36.0	58.3	2.6	1.7	1.15	14,190	1
Chilton	14	2.2	36.2	57.3	4.3	1.5	0.58	14,200	1
No. 2 Gas	15	1.9	38.8	56.6	2.7	1.4	0.89	14,450	1
Alma	16	2.1	37.7	53.1	7.1	1.5	1.78	13,810	1
Pratt	17	1.5	30.1	59.2	9.2	1.6	0.78	13,730	1
Pratt (washed)	18	3.0	31.5	62.4	3.1	1.7	0.60	14,510	1
20 % No. 7, 80 % No. 17	18A	3.4	28.2	62.2	6.2	1.6	1.2	14,030	1
Lower Sunnyside	19	4.6	38.8	50.6	6.0	1.5	1.04	13,030	1
Thick Freeport	20	[illegible].2	36.3	55.2	6.3	1.5	1.28	13,870	1
Green River	21	10.1	36.2	47.0	6.7	1.5	2.49	11,980	1
Pittsburgh	22	1.9	37.3	54.8	6.0	1.6	0.60	14,010	1
Pocahontas No. 4	23	0.8	15.4	79.0	4.8	1.1	0.47	14,870	1
80 % No. 22, 20 % No. 23	24	1.3	32.5	60.3	5.9	1.5	0.60	14,170	1
60 % No. 22, 40 % No. 23	25	1.4	28.3	64.8	5.5	1.5	0.57	14,330	1
Sewell	26	1.3	20.8	75.8	2.1	1.7	0.82	15,190	1
Sewell	27	1.9	26.5	69.2	2.4	1.6	0.47	14,970	1
Pittsburgh	28	1.8	35.1	57.7	5.4	1.6	0.93	14,140	1
80 % No. 28, 20 % No. 26	29	1.8	32.0	61.4	4.8	1.6	1.01	14,320	1
80 % No. 28, 20 % No. 27	30	1.9	33.6	59.8	4.7	1.7	0.88	14,330	1
Clintwood	31	1.9	31.5	61.0	5.6	1.8	0.89	14,350	2
Pittsburgh	32	2.0	41.3	48.7	8.0	1.2	4.44	13,320	3
Millers Creek	33	3.3	37.8	56.2	2.7	1.6	1.58	13,930	4
70 % No. 33, 30 % No. 23	34	3.3	31.0	62.3	3.4	1.4	1.14	13,980	4
30 % No. 33, 70 % No. 28	35	2.3	35.7	57.3	4.7	1.6	1.06	14,050	4
Alma	36	2.4	35.7	58.7	3.2	1.5	0.57	14,520	5
80 % No. 36, 20 % No. 23	37	1.4	32.0	63.0	3.6	1.5	0.55	14,640	5
70 % No. 36, 30 % No. 23	38	1.5	30.2	64.5	3.8	1.4	0.56	14,620	5
Upper Cedar Grove	39	1.8	36.2	55.2	6.8	1.4	1.46	13,920	5
80 % No. 39, 20 % No. 41	39A	1.6	32.5	59.7	6.2	1.2	1.29	14,130	5
70 % No. 39, 30 % No. 41	39B	1.4	30.8	62.2	5.6	1.3	1.08	14,310	5
Lower Cedar Grove	40	1.8	35.3	58.6	4.3	1.5	0.64	14,400	5
80 % No. 40, 20 % No. 41	40A	1.9	31.3	62.8	4.0	1.5	0.62	14,500	5
70 % No. 40, 30 % No. 41	40B	2.0	29.8	64.2	4.0	1.5	0.60	14,530	5
Beckley	41	2.3	17.9	76.0	3.8	1.4	0.43	14,820	5
20 % No. 41, 80 % No. 28	41A	1.6	32.3	61.2	4.9	1.6	0.98	14,320	5
30 % No. 41, 70 % No. 28	41B	1.5	30.5	63.1	4.9	1.6	0.88	14,390	5
Upper Banner	42	1.6	31.8	62.3	4.3	1.5	0.76	14,710	6
80 % No. 42, 20 % No. 41	42A	1.4	29.1	65.2	4.3	1.5	0.69	14,690	6
70 % No. 42, 30 % No. 41	42B	1.4	27.9	66.6	4.1	1.5	0.66	14,710	6
Dorothy	43	2.0	34.5	59.4	4.1	1.4	0.63	14,470	5
80 % No. 43, 20 % No. 41	43A	1.4	31.5	62.8	4.3	1.6	0.55	14,590	5
70 % No. 43, 30 % No. 41	43B	1.5	29.4	65.0	4.1	1.6	0.53	14,620	5
Powellton A	44	1.8	32.4	62.9	2.9	1.4	0.76	14,790	5
80 % No. 44, 20 % No. 41	44A	1.5	29.6	65.8	3.1	1.5	0.71	14,880	5
70 % No. 44, 30 % No. 41	44B	1.4	28.4	66.9	3.3	1.5	0.69	14,850	5
Indiana No. 4	45	13.7	32.7	46.2	7.4	1.3	0.89	11,490	6
80 % No. 45, 20 % No. 41	45A	8.5	30.6	54.1	6.8	1.5	0.74	12,560	6
70 % No. 45, 30 % No. 41	45B	10.3	28.8	54.5	6.4	1.5	0.91	12,380	6
Eagle	46	1.6	31.9	61.0	5.5	1.6	0.65	14,270	5
80 % No. 46, 20 % No. 41	46A	2.3	28.9	63.6	5.2	1.5	0.62	14,280	5
70 % No. 46, 30 % No. 41	46B	2.3	28.0	64.7	5.0	1.5	0.65	14,260	5
Lower Kittanning	47	1.7	16.3	75.3	6.7	1.4	1.26	14,310	7
20 % No. 47, 80 % No. 28	47A	1.7	30.6	62.0	5.7	1.5	1.12	14,070	7
30 % No. 47, 70 % No. 28	47B	1.9	29.0	63.3	5.8	1.6	1.24	14,060	7
Lower Kittanning (washed)	48	2.9	16.1	75.3	5.7	1.4	0.77	14,300	7
20 % No. 48, 80 % No. 28	48A	1.7	31.2	61.8	5.3	1.5	0.95	14,180	7
30 % No. 48, 70 % No. 28	48B	1.8	29.2	63.6	5.4	1.5	0.96	14,100	7
"Puritan"	49	19.7	29.4	46.4	4.5	1.3	0.40	10,500	8
Upper Kittanning	50	1.8	16.9	72.0	9.3	1.2	2.30	13,860	7
20 % No. 50, 80 % No. 28	50A	1.8	31.8	60.1	6.3	1.5	1.18	14,080	7
30 % No. 50, 70 % No. 28	50B	2.3	30.0	61.2	6.4	1.5	1.25	14,000	7
Upper Kittanning (washed)	51	2.2	17.3	72.0	8.5	1.4	1.18	13,980	7
20 % No. 51, 80 % No. 28	51A	1.9	30.9	61.2	6.0	1.6	1.06	14,010	7
30 % No. 51, 70 % No. 28	51B	1.6	29.4	62.5	6.5	1.6	1.07	14,050	7

TABLE I.—(*Continued*)

Bed	Coal No.	Moisture	Volatile Matter	Fixed Carbon	Ash	Nitrogen	Total Sulphur	B.t.u. as Received	Ref. No.
Pittsburgh	52	2.4	36.6	53.2	7.8	1.5	1.60	13,560	9
80 % No. 52, 20 % No. 41	52A	2.0	33.0	58.1	6.9	1.5	1.30	13,870	9
70 % No. 52, 30 % No. 41	52B	2.3	31.2	59.9	6.6	1.5	1.25	13,880	9
Pond Creek	53	2.7	31.9	61.1	4.3	1.5	0.74	14,290	10
80 % No. 53, 20 % No. 41	53A	2.3	29.7	63.6	4.4	1.4	0.71	14,390	10
70 % No. 53, 30 % No. 41	53B	2.0	28.4	65.3	4.3	1.4	0.62	14,470	10
High Splint	54	3.3	37.5	55.5	3.7	1.4	0.54	13,870	11
80 % No. 54, 20 % No. 41	54A	2.7	34.0	59.5	3.8	1.4	0.54	14,070	11
70 % No. 54, 30 % No. 41	54B	2.7	32.0	61.6	3.7	1.4	0.51	14,080	11
Sewell	55	1.6	21.8	73.7	2.9	1.4	0.72	14,950	12
20 % No. 55, 80 % No. a36	55A	1.9	34.6	59.5	4.0	1.4	0.51	14,550	12
30 % No. 55, 70 % No. a36	55B	1.9	32.9	61.4	3.8	1.5	0.60	14,570	12

Since the tests were made in duplicate, the information necessary for calculating the errors[14] in the measured quantities was obtained, but unfortunately only a single value for each quantity was reported in the published results. This means that it is not possible to state accurately whether or not the equations fit the data within the limits of experimental error. In making such judgments it has been necessary to rely on estimates of the errors obtained from other sources and on general knowledge of the processes involved.

In studying the correlation of the assay-test data with coal analyses various possible relations are suggested by the known facts about coal carbonization and by examination of the data. It is necessary: (1) to determine whether there is evidence for the existence of any suggested relation, and if so, to give a numerical measure of how closely the relation holds; and (2) to determine the best equation expressing the relation and to calculate the error involved in using this equation to compute the results of assay tests. For these purposes it is necessary to use statistical methods. The details of these methods will not be discussed here because they are readily available in the texts on the subject, of which two may be mentioned.[15]

If two quantities are linearly related, the *correlation coefficient* gives a numerical measure of the closeness of this relationship. A value of the correlation coefficient near zero indicates practically no relation and the nearer the correlation coefficient is to unity the more exactly the relation holds. The *multiple correlation coefficient* is a similar measure for the relation between a dependent variable and more than one independent variable. In the tables, the same symbol, R, is used for the simple and the multiple correlation coefficient; the multiple correlation coefficient being used whenever there is more than one independent quantity.

The *probable errors* given for the equations in this paper measure the errors involved in using the equations to compute assay-test results. Half the number of the values calculated from an equation differ from the corresponding observed values by less than the probable error. The probable error is approximately equal to 0.845 times the average error, where average error means that the errors are averaged without regard to algebraic sign.

Least squares have been used to obtain most of the equations given here. This method reduces the sum of the squares of the differences between the values calculated from the equation and the observed values to a minimum for an equation of the given form.

The term "significant" is used in this paper in a statistical sense, meaning that the significance or nonsignificance was decided by the appropriate statistical test. For example, at 800°, 900°, and 1000° data are available for both the 13-in. and 18-in. retort sizes. It was necessary to decide

whether these data should be treated separately or the data for the two sizes should be taken together. In each case this was decided by calculating the differences between 13-in. and 18-in. retort values for coals that were tested in both, and applying the statistical test for the significance of these differences. Similarly, when it is stated that a certain coal property has no significant effect on one of the assay-test results, it is meant that the statistical tests show that there is no evidence of a significant effect.

Most of the correlating equations developed in this paper are linear. This does not imply that the "true" relations are linear rather than of some more complicated form. Linear equations are preferred mainly because they are simpler: to derive, to test for significance, and to use for calculating carbonizing properties from coal analyses. A more complicated equation is justified only if the data definitely indicate that the linear form is incorrect. The linear equations are to be regarded as approximations, which may need to be replaced when more complete or more accurate data become available.

Most of the equations were derived first for each temperature separately, then, if possible, these separate equations were combined to give a single equation that would include the effect of temperature. Where this was done both the separate equations and the temperature-dependent equation have been given. The two equations give values for the calculated quantities that differ only very slightly; however, the temperature-dependent equations are to be preferred since they are based on a correlation of the data at several temperatures rather than at a single temperature.

It is necessary to point out that the equations stated in this paper apply only to the results of carbonizations in the cylindrical retorts used in the assay tests, and cannot be used directly to calculate carbonization results in commercial by-product ovens. It may be true that some of the equations may apply to by-product ovens; for example, sulphur in coke, as stated below. However, additional data are needed before the equations can be taken over directly. It also seems probable that relations of the same *form* would hold for by-product coke ovens, and if the data were available the correct equations could be derived.

Other authors[16] have published correlations of some of the data obtained in the BM-AGA assay tests, which only partly, however, overlap the material considered in this paper. These will not be discussed here, but it is worth pointing out that these correlations have involved more complicated relations than we have found necessary and, what is more important, there are few statements as to the agreement between the equations obtained and the observed values, so that it is not possible to say (without a great deal of additional work) whether the correlations are satisfactory. It is also to be noted that the correlations given here, with the exception only of those referring to the elementary composition, are based on the *proximate* analysis of the coal only. It was not necessary to use ultimate analyses to obtain fairly satisfactory equations for the coke and by-product yields and properties.

Carbonizing Time

In making an analysis of the kind attempted herein it is important to be assured that each coal carbonized at a given temperature has been given time to reach approximately the same degree of completion of the decomposition reactions characteristic of that temperature. The criterion used by the Bureau of Mines was stated usually to be the rate of gas evolution (ref. 13, p. 96). Only for the first 30 coals are numerical values given. It is unfortunate that these values of carbonizing time do not agree with those read from the curves given as illustrative (ref. 1, Table 14, and Fig. 9; ref. 13, Fig. 47). This

may be attributed to the fact that the curves are for a single sample while the numerical values are averages of at least

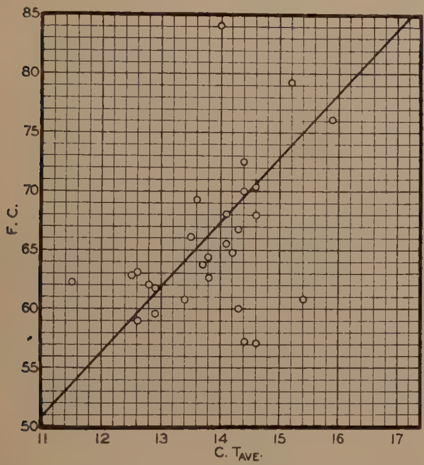

FIG. 1.—PLOT OF AVERAGE CARBONIZING TIME AGAINST FIXED CARBON, DRY, MINERAL-MATTER-FREE BASIS (AFTER FIELDNER AND DAVIS[1]).

two experiments. However, this circumstance renders it unwise to use the data read from curves for coals 31 to 55 for comparison with the numerical data for the first 30 coals (ref. 1, Table 14).

It was shown in *Monograph* 5 (ref. 1, Fig. 14) that the average carbonizing time was not closely related to the percentage of dry, mineral-matter-free fixed carbon of the coal, as is evident in Fig. 1, reproduced from the original publication. Other considerations would lead to the conclusion that the time to carbonize a given charge of coal would depend both on the yield of coke finally obtained and the amount of moisture to be evaporated. As will be shown later, the yield of coke is related directly to per cent of fixed carbon plus ash in the coal. The conclusion stated above assumes that the carbonizing time is directly related to the amount of heat needed for carbonization and that this latter value, as a first approximation, is determined by the heat in the final product plus the heat used in vaporization of water. This assumption may be expressed as an equation relating carbonizing time (CT) to fixed carbon plus ash ($FC + A_C$) and to moisture (H_2O) of

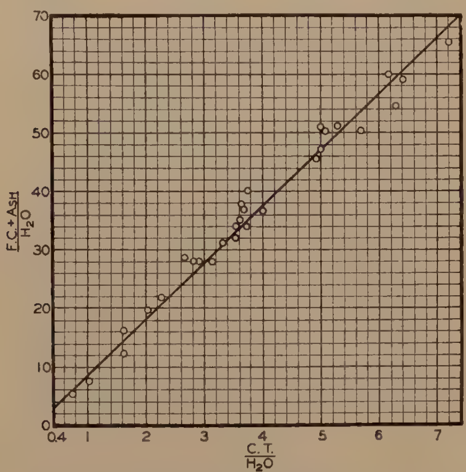

FIG. 2.—PLOT OF CARBONIZING TIME AT 900°C. ACCORDING TO EQUATION 1.

$$CT = a(FC + A_C) + b(H_2O) \qquad [1]$$

the form

$$CT = a(FC + A_C) + b(H_2O) \qquad [1]$$

The improvement by this method of representing the data is shown graphically in Fig. 2, where the ratio of $CT/(H_2O)$ is plotted against the ratio of

$$\frac{(FC + A_C)}{(H_2O)}$$

which, according to equation 1, should give a straight line, and which is closely approximated.

Before giving the results of the analysis of the data on this basis, certain conflicts between the assumptions made and actual experimental procedure must be mentioned. A basic assumption that is not accurately fulfilled is that the weight of coal plus moisture charged to the retort is constant and does not vary from coal to coal. However, the "normal" charge density (ref. 1, p. 88) is given as 50 to 53 lb. per cubic foot; this density is also stated

to be 50 ± 2 lb. per cubic foot (ref. 1, p. 137). It is also stated that, for coals with volatile matter > 28 per cent, 25 in. of the 26-in. height is filled but for coals with volatile matter < 28 per cent, only 23 to 24 in. are filled. These facts introduce a variability of 12 per cent in the charge of coal carbonized and may be expected to affect the calculated results. A similar estimate of the variability of the charge is given by the statement (ref. 1, p. 15) that the retort is "13 in. in diameter and 26 in. high, holding 85 to 100 lb."; i.e., 92.5 lb. ±8 per cent. Further, it is stated (ref. 1, p. 88) that about 15 per cent less than normal charge was used with low-volatile and medium-volatile coals, but when this procedure was introduced into the series of tests is not specifically stated. These points are brought out here not in criticism of the work, but to emphasize that at least a part of the failure to reproduce accurately the observed test data by calculation from analysis may be due to inherent inadequacy of the basic assumptions made. It is, from this point of view, unfortunate that the weight of the coal charge in each experiment is not given.

In Table 2 are given the results of the analysis of the carbonizing-time data for the first 30 coals in a 13-in. retort using a least-squares method: T is the carbonizing temperature; n, the number of coals; R, the correlation coefficient; PE, the probable error in per cent; and E_{max}, the maximum positive and negative errors, i.e., calculated − observed. Only two coals give errors of only one sign for all temperatures: one of these is the low-volatile coal No. 23, for which a correction of 15 per cent might be applied as indicated in the preceding paragraph, and which then would not be "abnormal" in behavior.

TABLE 2.—*Carbonizing Time, 13-inch Retort*

$$CT = a(FC + Ac) + b(H_2O) \quad [1]$$

T., C.	500°	600°	700°	800°	900°	1000°	1100°
n	28	30	28	30	30	30	29
CT, avg., hr	32.5	23.2	14.9	9.9	7.0	5.4	4.2
CT, max., hr	38.8	25.4	16.5	11.3	8.8	6.4	5.1
CT, min., hr	24.8	18.0	12.2	8.8	5.6	4.4	3.6
R	0.69	0.79	0.72	0.73	0.77	0.70	0.67
a	0.453	0.326	0.209	0.140	0.098	0.076_5	0.059_4
b	1.061	0.764	0.486	0.282	0.248	0.151	0.140
$PE_{[1]}$, per cent	5.9	4.6	5.1	4.1	4.7	4.6	5.9
E, max., per cent	+26.0	+20.0	+13.6	+16.3	+20.0	+18.2	+34.2
E, max., per cent	−11.9	−11.6	−15.0	−11.5	−12.0	−13.8	−13.3
$\frac{100b}{CT\text{, avg}}$	3.26	3.29	3.26	2.85	3.54	2.80	3.33
a'	(0.630)	0.305	0.209	0.141	0.100	0.076_6	0.060_4

Further analysis of the data indicates that the coefficient of moisture b is essentially a constant percentage of the average carbonizing time as shown in the row $100b/CT_{avg}$, and that on the average each per cent of moisture—assuming constant fixed carbon plus ash—increases the carbonizing time 3.19 per cent. This indicates the magnitude of the effect of analytical errors on the calculated results, for if the probable error of the moisture determination were ±0.5 per cent, the probable error so introduced into the calculated carbonizing time would be ±1.6 per cent.

Excluding the value for 500°, the coefficient of the fixed carbon plus ash is related to the carbonizing temperature by the relation

$$\log a = -2.53 + \frac{1800}{T + 273} \quad [2]$$

The values calculated from this equation are given in Table 2 as a'. It will be seen

in the following discussion that many of the coefficients have the same general dependence on temperature. This fact lends confidence to the analysis of the data, since in general constants of chemical transformations also show this dependence. If this relation were used to calculate the carbonizing times at 500°, it would be found that the observed carbonizing times were all about 33 per cent too low. This is of particular interest, since the rate of gas evolution could not be used as the criterion for determining experimentally the end of the carbonization at 500° and the experiments were terminated more or less arbitrarily. Many of the other data of the carbonization assay indicate also that the 500° carbonizations were not carried to a point comparable with the carbonizations made at higher temperatures.

The analyses of the carbonizing-time data at individual temperatures and the determination of the dependence of the resulting coefficient on temperature permit, by a method involving approximations, formulation of a single equation to reproduce all the data as follows:

$$\log \frac{CT(1 - 0.032\mathrm{H_2O})}{a''(FC + A_C)} = \frac{1800}{T + 273} \quad [3]$$

In this equation, the constant a'' is believed to be the only constant dependent on the apparatus. The equation has been used with success in calculating the effects of changes in coal analysis and carbonization temperature on carbonizing time as reported in other studies, determining a'' from a single observation only.

Relation of Carbonizing Time to Retort Diameter

If a charge of coal could be regarded simply as a heat-conducting solid, simple theory would demand that the carbonizing time for a given coal should be proportional to the square of the radius of the retort. In certain cases, this relationship has been said to be satisfied by experimental data.[17] At 900°C. and at 1000°C., numerical data are given (ref. 1, Table 14) for the carbonizing times in both 13-in. and 18-in. retorts for coals No. 9 and Nos. 13 to 30, which can be used to check the "square" law. Because of the uncertainty of the graphical data, discussed previously, these data for the later coals were not used in determining the relation of carbonizing time to retort diameter. Analysis of the data shows that the ratio of the carbonizing time in the 18-in. retort to that in the 13-in. retort ranges at 900° from 1.51 to 1.83 with an average of 1.66, which has a probable error of 0.06; and at 1000° from 1.42 to 1.85 with an average of 1.64, which has a probable error of 0.07. The square law demands a ratio of 1.92. The observed ratio of 1.65 ± 0.07 shows that the carbonizing time for a given coal is proportional to an exponent less than the square of the radius of the retort; namely, 1.54 ± 0.12.

Dependence of Carbonizing Time on Density of Charge

If the assumptions made for calculating the carbonizing time of a given coal from its proximate analysis were correct, a change in the density of the charge in the retort would cause a proportional change in the carbonizing time. Data are given only for coal No. 16 carbonized at 900°C. with a "normal" density of 50 lb. per cubic foot and with densities of 56 and 59 pounds per cubic foot. It is unfortunate that coal No. 16 gives the second largest positive error for the calculated carbonizing time at 900° and normal charge density. However, the agreement between calculated and observed values improves for the higher charge densities, as follows: at charge densities of 50, 56, and 59 lb. per cubic foot, the observed carbonizing times are 5.6, 6.8, and 6.9 hr., respectively, and the calculated values are 6.4, 7.2, and 7.5 hr., respectively. The errors in the calculated

values are no greater than expected from the probable error of 4.7 per cent as given in Table 2.

Coke

Yield

Both in the carbonization assay and in the proximate analysis of coals, the volatile matter and moisture are driven off, leaving a residue, which is the sum of fixed carbon plus ash in the proximate analysis. Even though the coke is prepared under different conditions of temperature and rate of heating, both of which must be rigorously controlled in the analytical procedure to obtain satisfactory results, nevertheless a close relation between the coke yield at any given temperature and the sum of fixed carbon plus ash is to be expected. The correlation is very high ranging from 0.983 to 0.997 (Table 3).

In analyzing the data on coke yield, it was determined that there was no significant difference between the results obtained in the 13-in. retort and in the 18-in. retort. Where both were given, an average was taken and considered as a single observation. The yield of coke, K, is related to the fixed carbon and ash by the equation:

$$K = a + b(FC + Ac) \qquad [4]$$

Preliminary analysis showed that the data for coals No. 45, Indiana No. 4 seam, and No. 49, a Colorado subbituminous coal, were not consistent with the data for the other coals. Therefore, the coke yields of these two coals were not used in determining the constants and the errors given in Table 3. The constants a' and b' are those determined directly from the data by the method of least squares. Neither a' nor b' shows a regular dependence on temperature and the variation of b' is small enough to suggest that it could be replaced by a constant b independent of temperature. When this is done the values of a' must be changed to the values a given. The errors given in the table are those from using the equation with the constants a and b. It should be noted that it follows from the values of a that the coke yield is linearly dependent on temperature from 800° to 1100°, decreasing 0.1 per cent for each increase of 100°C. in carbonizing temperature.

Examination of the errors involved in calculating coke yield of the individual coals indicates only that equation 4 is probably an oversimplification. Two lower-rank coals, besides Nos. 45 and 49, consistently give maximum errors; the Illinois No. 6 seam coal, No. 10, and the Lower Sunnyside coal, No. 19 (Table 1). There are insufficient data for the lower-rank coals to justify formulation of a curvilinear relationship, which might give better agreement between calculated and observed coke yields. It might be mentioned that the

TABLE 3.—*Coke Yield, K*

$K = a + b(FC + Ac)$ [4]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	48	50	49	52	83	49	30
K, avg., per cent	79.4	75.5	73.0	72.0	72.1	71.9	71.0
K, max., per cent	93.0	89.6	87.3	86.0	85.8	86.0	86.7
K, min., per cent	68.8	64.4	62.0	60.9	60.6	60.8	60.4
R	0.983	0.992	0.997	0.996	0.986	0.987	0.985
a'	29.4	21.8	20.0	19.5	19.0	20.0	16.2
b'	0.748	0.802	0.794	0.788	0.793	0.778	0.832
a	26.8	22.8	20.3	19.3	19.2	19.1	19.0
b	0.791	0.791	0.791	0.791	0.791	0.791	0.791
$PE_{[4]}$, per cent	0.49	0.42	0.37	0.39	0.46	0.60	0.73
E, max., per cent	+1.8	+1.5	+1.2	+1.8	+2.7	+3.1	+3.7
E, max., per cent	−1.8	−1.2	−0.9	−1.0	−1.8	−1.8	−1.2

relationship

$$K = 1.0763(FC + A_C)_{\text{dry basis}} \quad [5]$$

was proposed[9] as one that satisfactorily reproduced the observed data at 900° for the six Pittsburgh seam coals. Eq. 4 reproduces the data with a probable error of only 0.8 that of Eq. 5.

TABLE 4.—*Hydrogen Content of Cokes*, $[H]_K$

$\log [H]_K = 1.420 - 0.00173T$ [6]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	50	52	51	68	122	70	27
Avg. $[H]_K$ obs., per cent	3.41	2.59	1.62	1.16	0.73	0.50	0.31
PE, per cent	0.09	0.10	0.11	0.10	0.08	0.09	0.07
$[H]_{K'}$ obs., per cent				1.00	0.76	0.49	0.29
$[H]_K$ cal., per cent	3.59	2.41	1.62	1.09	0.73	0.49	0.33
E, max., per cent	+0.49	+0.11	+0.22	+0.19	+0.23	+0.19	+0.13
E, max., per cent	−0.21	−0.49	−0.38	−0.61	−0.47	−0.41	−0.37

Composition

Hydrogen Content.—Previous studies[18] have shown that on an ash and moisture-free basis the hydrogen content of carbonaceous materials heated to 800° and above is primarily dependent on the maximum temperature of carbonization. In Table 4 there are given the average hydrogen contents $[H]_{K_{obs}}$ of the cokes prepared in the BM-AGA survey and the probable error of this average. The figures in the row marked $[H]_{K'_{obs}}$ at 800° to 1100° are values obtained in another study of anthracite cokes.[18] The hydrogen contents of the cokes prepared from coals 2, 3 and 11 at 1100°C. were not included in the average value given for this temperature. It may be noted that the greatest decrease in the observed hydrogen content is between 600° and 700° and this may be associated with the well-known fact that this temperature range is "critical"[19] for the dependence of many properties of "carbon" on temperature.

The calculated values were obtained from the equation

$$\log [H]_K = 1.420 - 0.00173T \quad [6]$$

which was not determined by least-squares analysis of the data, but was adjusted to give the best fit at 900° and 1000°, the temperatures at which there are the greatest number of observations. The maximum positive and negative errors of the calculated values are also given in Table 4. The fact that the maximum negative errors are generally much greater than the maximum positive errors, which means that the observed values are more frequently greater than the calculated values, may be justified on the grounds that the errors in determination of hydrogen tend usually to give values too high, probably because of occlusion of moisture by the sample or by the combustion tube filling.

Volatile Matter.—It is generally recognized that the volatile matter of cokes, like the hydrogen content, is primarily dependent on temperature of carbonization. The average percentage of volatile matter in the cokes and its probable error are given in Table 5. The volatile matter remaining in the 500° cokes is not independent of the volatile-matter content of the original coal, but is represented by the following equation:

$$(VM)_{K_{500}} = 6.92_5 + 0.1_{00}(VM)_C \quad [7]$$

Also, it appears unreasonable that there would remain true volatile matter in a coke prepared at 1000° or 1100° which could be removed by subsequent heating to 950° for 7 min. In deriving an equation, therefore, to express the dependence of the percentage of volatile matter in coke as a

function of temperature, the observed averages at 500°, 1000°, and 1100° were not used. The equation is:

$$\log (VM) = 2.391 - 0.00265T \quad [8]$$

and reproduces the data within the maximum positive and negative errors shown in Table 5. The fact that the volatile-matter content at 1100° is about equal to the hydrogen content of the coke is in accord with the observation of Lowry and Hulett[20] that on heating carbons from 1100° to 1400°C. in vacuo the gas liberated was essentially pure hydrogen. Here also the observed volatile-matter content is generally greater than that calculated for the cokes prepared at 800° and above, and may be attributed to the greater likelihood that errors associated with its determination will give values that are too great rather than too small.

TABLE 5.—*Volatile Matter Content of Cokes,* $(VM)_K$

$\log (VM)_K = 2.391 - 0.00265T$ [8]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	50	52	51	68	122	70	30
Avg. $(VM)_K$ obs., per cent	10.30	6.25	3.52	1.92	0.94	0.87	0.84
PE, per cent	0.89	0.63	0.55	0.47	0.35	0.28	0.28
$(VM)_K$ cal., per cent		6.32	3.44	1.87	1.01	(0.55)	(0.30)
E, max., per cent		+2.12	+1.74	+1.17	+0.81	+0.25	
E, max., per cent		−1.88	−1.36	−1.83	−1.69	−1.75	−1.90

TABLE 6.—*Nitrogen Content of Cokes,* N_K

$N_K' = N_C' + D$ [9]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	50	52	51	68	122	70	30
Avg. N_K obs., per cent	1.86	1.77	1.61	1.56	1.50	1.30	0.95
D	0.22	0.12	−0.04	−0.06	−0.12	−0.35	−0.74
$PE_{[9]}$, per cent	0.08	0.09	0.08	0.08	0.08	0.11	0.13
E, max., per cent	+0.22	+0.62	+0.26	+0.64	+0.68	+0.45	+0.46
E, max., per cent	−0.28	−0.28	−0.24	−0.26	−0.32	−0.35	−0.44
D_U	0.23	0.13	−0.02	−0.04	−0.10	−0.30	−0.66

Nitrogen Content.—The percentage of nitrogen in the dry, ash-free coke, N_K', is related to the percentage of nitrogen in the dry, mineral-matter-free coal, N_C', by the equation:

$$N_K' = N_C' + D \quad [9]$$

where *D* is a temperature-dependent constant given in Table 6. In this table there are also given the average percentage of nitrogen in the cokes and the probable, as well as the maximum positive and negative, errors of the calculated values. Here, as also for the hydrogen content of the cokes, the probable error is essentially independent of the temperature at which the cokes were prepared and may be attributed to analytical errors. This relationship between the percentage of nitrogen in the coke and that in the coal is important in calculating the yield of ammonium sulphate obtained in the carbonization assay, as will be shown later; for this purpose, and in order to simplify calculation, a new constant, D_U, was calculated, and is included in Table 6, which may replace *D* if the percentage of nitrogen in the coke is on the dry basis and the percentage of nitrogen in the coal is on the as-carbonized basis.

Sulphur Content.—Examination of the data on sulphur content of the 443 cokes prepared from the different coals showed that the percentage of sulphur was independent of both retort size and carbonization temperature. The percentage of sulphur in the dry, ash-free coke, S_K', may be calculated by the use of the equation:

$$S_K' = 0.85S_{org} + 0.86S_{pyr} \qquad [10]$$

where S_{org} is the percentage of organic sulphur in the dry, mineral-matter-free coal, and S_{pyr} is the percentage of pyritic sulphur, with a probable error of ± 0.068 per cent. On these bases, the sulphur in the experimental cokes ranged from 0.47 to 4.00 per cent, the organic sulphur in the coals from 0.32 to 2.63 per cent, and the pyritic sulphur from 0.01 to 2.00 per cent. Since the coefficients of S_{org} and S_{pyr} are so nearly equal, and these data are not always available, and since the correlation coefficient between the sulphur in coke on a dry basis, S_K, and the total sulphur in the coal as carbonized, S_T, was so high (0.98), the following simpler equation

$$S_K = 0.084 + 0.759S_T \qquad [11]$$

is to be preferred, especially since its use gives a slightly smaller probable error than does Eq. 10; i.e., ± 0.065 per cent. On the dry basis the sulphur in the coke averages 0.83 per cent and ranges from 0.4 to 3.6 per cent. The maximum positive error in the use of Eq. 11 was 0.33 per cent and the maximum negative error was −0.36 per cent.

It may be of interest that the Coke Chemists Committee of one steel company has adopted the use of Eq. 11 in place of the analytical determination previously used. Further, the equation has been used to calculate the sulphur in 151 daily samples of coke from another steel company ranging in sulphur from 0.49 to 0.64 and the calculated values showed a probable error of only 0.023 per cent.

Table 7.—*Oxygen Content of Cokes*, $[O]_K$

$$[O]_K = 3740/T - 3.84 \qquad [13]$$

$$\log [O]_K = 1.674 - 0.00215T \qquad [14]$$

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	50	52	51	68	122	70	30
Avg. $[O]_K$ obs., per cent	3.66	2.36	1.54	0.87	0.29	0.30	0.36
PE, per cent	0.49	0.34	0.28	0.27	0.17	0.18	0.24
$[O]_K$ cal.[13], per cent	3.64	2.39	1.50	0.84	0.32		
$[O]_K$ cal.[14], per cent	(3.97)	2.42	1.48	0.90	(0.55)	(0.33)	(0.20)
E, max.[13], per cent	+1.44	+1.79	+1.20	+0.84	+0.32		
E, max.[13], per cent	−1.66	−1.01	−0.70	−1.06	−0.98	−1.10	−1.20

Oxygen Content.—Since the determination of oxygen in coke is by difference and therefore includes the cumulative errors of the directly determined elements, and since in any case the percentage of oxygen in high-temperature coke is small, it might well be considered that statistical treatment of the data is not warranted. The assumption was made, however, that the average percentage of oxygen in cokes, on a dry, ash-free basis, prepared at a given temperature was probably the correct one for all cokes. This average observed value, $[O]_{K_{obs}}$, is shown Table 7 together with the probable error of the deviations from the average, which is surprisingly small. The oxygen content of the 500° cokes is not independent of the percentage of oxygen of the parent coal, $[O]_C$, but may be calculated approximately by the equation:

$$[O]_{K_{500}} = 2.0 + 0.25[O]_C \qquad [12]$$

It seems unreasonable that the percentage

of oxygen in the coke should increase above 900° as the data in Table 7 show, and this observed fact also makes the average value at 900° itself uncertain.* Therefore, in attempting to find the relation between oxygen content and carbonization temperature, the three values at 600°, 700° and 800° must be given the greatest weight. Six different forms of equations have been fitted to the data. The best agreement is given by the following:

$$[O]_K = 3740/T - 3.84 \qquad [13]$$

and this equation reproduces both the 500° and the 900° observed averages. It requires that the oxygen content of cokes become zero at 974°C. From various chemical considerations it is preferable to have an equation of a form that shows that the oxygen content decreases logarithmically with temperature, as in the following equation:

$$\log [O]_K = 1.674 - 0.00215T \qquad [14]$$

which reproduces the most reliable values (cf. supra) satisfactorily. Inclusion of a third term improves the agreement, particularly at 900°, but probably is not justified by the data. The oxygen contents calculated by these equations and the maximum positive and negative errors resulting from the use of Eq. 13 are included in Table 7.

General Considerations.—It would seem, from this analysis of the data obtained by the Bureau of Mines in carbonization assay tests, that the chemical composition of cokes is primarily dependent on temperature. Only for sulphur is temperature not important, and, except for sulphur and nitrogen, the composition is independent of the analysis of original coal at temperatures above 500°C. These conclusions have several important consequences.

If the analysis of the coal carbonized and the temperature of carbonization are known, the chemical composition of the coke with respect to hydrogen content, volatile-matter content, nitrogen content, sulphur content and oxygen content can be calculated with an error probably not exceeding that involved in its experimental determination. Also, even if neither coal analysis nor carbonization temperature is known, the latter can be calculated from the coke analysis. For this purpose, the hydrogen content is of greatest value, but it must be emphasized that extreme precautions should be taken that, in the analysis for hydrogen, errors due to occluded moisture in the coke and in the combustion-tube filling are minimized. Then Eq. 6 may be rewritten:

$$T\ °\text{C.} = 821 - 578 \log [H]_K \qquad [15]$$

or

$$T\ °\text{F.} = 1510 - 1040 \log [H]_K \qquad [16]$$

Experience with the use of hydrogen content of carbonaceous materials heated to high temperatures in the past 15 years has led to such confidence that in the Coal Research Laboratory it is accepted as a "hydrogen thermometer."

The dependence on temperature of the sulphur, oxygen, nitrogen, and hydrogen contents of cokes may be used to help in the understanding of the chemical nature of coke. As coal is heated, X-rays show an increase in graphitization, the crystallites increasing in size with increasing temperature,[21] as is also generally true for so-called amorphous carbons.[22] Graphite is essentially a six-carbon ring system extended indefinitely in two dimensions, the planes having definite relations in the third dimension. That the sulphur content of coke made from a given coal is independent of temperature from 500° to 1100°, in amounts up to about 4 per cent, suggests that below 500° the sulphur is all "fixed" in six-membered rings replacing a carbon in the normal graphite structure. The extreme temperature stability of the carbon-sulphur complex is well known (see, for

* It should be observed that 20 coals show zero oxygen content at 900°; 15 coals at 1000°, and one coal at 1100°.

example, ref. 23). The carbon-oxygen complex is not so stable and its thermal decomposition has been studied by several investigators (see, for example, refs. 20, 24). There is evidence that the nitrogen in coals of bituminous and higher ranks is largely in heterocyclic rings,[25] which, however, according to the coke analyses are not as stable as the rings containing sulphur. Hydrogen can be held in a graphitic structure only on the periphery of the crystallites; that significant amounts are held to temperature even as high as 1500°[18] has been explained by a combination of the unsaturation of peripheral carbon atoms and by the lack of mobility of the carbon atoms. As the temperature is increased, increasing numbers of carbon atoms, however, are enabled to rearrange their position into the stable graphite lattice and thereby release the hydrogen atoms previously attached to them.

From a chemist's point of view equations 6, 8 and 14, relating hydrogen, volatile matter, and oxygen contents to temperature, are not entirely satisfactory, since constants relating to molecular transformations are generally related linearly to the reciprocal of the absolute temperatures rather than to the centigrade temperature. It can be shown that the hydrogen content can be related to temperature, at temperatures from 600° to 1500°, by an equation of the form:

$$[\mathrm{H}] = a \sinh \frac{b}{T + 273} - c \qquad [17]$$

which is similar to the form that has been shown by Eyring[26] to describe the temperature dependence of viscosity of liquids. This fact is in agreement with the mechanism proposed in the preceding paragraph for the effect of temperature on the hydrogen content of cokes. Analogous equations undoubtedly could be fitted to the data for volatile matter and oxygen contents of coke, but at present do not appear to be justified.

Physical Properties

Calculation of the physical properties of the assay cokes at the different carbonization temperatures is generally more involved than the calculation of coke composition. Equations have been derived, which, it is felt, satisfactorily reproduce the data on apparent and true specific gravities, shatter-test data, and the percentage retained on ¼-in. screen in the tumbler test. No attempt was made to derive an equation for calculating porosity from coal analysis, since the porosity is a derived quality calculated from two observed properties, the apparent and true specific gravities.

Apparent Specific Gravity.—As a first approximation it was assumed that the apparent specific gravity of a coke, R_K, prepared at a given temperature would be determined primarily by its ash content, which in turn is determined by the ash content of the parent coal, A_C, multiplied by the factor $(100/K)$ where K is coke yield calculated from Eq. 4, and by the nature of the organic matter of the parent coal, which might be indicated by the fixed carbon on a dry ash-free basis, FC^*. Statistical analysis showed that the moisture content of the original coal also was a significant factor; provisionally this may be attributed to the controlling effect of moisture on the rate of heating up to and through the plastic range.[27] The equation is:

$$R_K = a + bFC^* + c(\mathrm{H_2O}) + dA_C(100/K) \qquad [18]$$

In determining the coefficients, analysis of the data showed that difference between the value of R_K for cokes prepared in the 13-in. and 18-in. retorts was not significant; where both were given an average was taken and counted as a single observation. As shown in Table 8, the correlation coefficients are not high, but probably this is attributable to the fact that the range of

values is not very great in comparison to the experimental error of the determination. The probable errors and maximum positive and negative errors show that the equation reproduces the data satisfactorily. The temperature dependence of the coefficients is irregular and no attempt was made to find expressions for it. Again, it should be emphasized that the coefficients probably depend on the carbonizing equipment and apply only to the BM-AGA assay data.

"True" Specific Gravity.—Since the composition of coke, as shown by the analysis given in preceding paragraphs, is dependent primarily on the temperature, it should be expected that the true specific gravity of cokes prepared at a given temperature would depend only on their ash contents. Analysis of the data, again averaging the values for cokes prepared in 13-in. and 18-in. retorts, shows that this is essentially true for the cokes prepared at 500° to 900°. However, it appears that for the cokes prepared at 1000° and 1100° the true specific gravity is determined in part also by the moisture content of the parent coal and by the content of dry, ash-free fixed carbon. The equation developed for calculating the true specific gravity of the cokes, V_K, is:

$$V_K = a + bA_C(100/K) + c(H_2O) + dFC^* \qquad [19]$$

In this equation the symbols have the significance assigned them previously, K being the yield of coke calculated from Eq. 4. The values of the coefficients and the errors of the calculated values are given in Table 9. It should be noted that the two-constant equation should be used at

TABLE 8.—*Apparent Specific Gravity of Cokes, R_K*

$R_K = a + bFC^* + c(H_2O) + dA_C(100/K)$ [18]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	42	50	49	52	83	49	30
R_K avg	0.73	0.75	0.79	0.83	0.85	0.85	0.85
R_K max	0.83	0.86	0.88	0.88	0.95	0.96	0.96
R_K min	0.61	0.68	0.73	0.70	0.76	0.77	0.77
R	0.52	0.65	0.53	0.56	0.56	0.60	0.76
a	0.553	0.566	0.695	0.776	0.773	0.773	0.653
100b	0.257	0.252	0.163	0.102	0.159	0.139	0.297
100c	−0.69	−0.76	−1.05	−1.65	−1.65	−1.49	−1.40
100d	0.38	0.46	0.21	0.29	0.14	0.27	0.45
$PE_{[18]}$	0.025	0.021	0.025	0.028	0.025	0.025	0.022
E, max., per cent	+0.10	+0.07	+0.09	+0.11	+0.08	+0.10	+0.07
E, max., per cent	−0.10	−0.08	−0.08	−0.11	−0.07	−0.10	−0.06

TABLE 9.—*"True" Specific Gravities of Cokes, V_K*

$V_K = a + bA_C(100/K) + c(H_2O) + dFC^*$ [19]

T.,C.	500°	600°	700°	800°	900°		1000°	1100°
n	48	50	49	52	83		49	30
V_K avg	1.46	1.57	1.74	1.84	1.88		1.89	1.93
V_K max	1.55	1.64	1.84	1.92	1.98		2.01	2.05
V_K min	1.41	1.52	1.69	1.77	1.79		1.79	1.83
R	0.77	0.46	0.42	0.72	0.74	0.75	0.87	0.88
a	1.422	1.543	1.708	1.796	1.817	1.752	1.621	1.669
100b	0.60	0.33	0.38	0.57	0.79	0.81	0.83	0.76
100c						0.10	−0.71	−1.35
100d						0.09	0.34	0.35
$PE_{[19]}$	0.011	0.015	0.020	0.013	0.015	0.015	0.015	0.017
E, max	+0.04	+0.04	+0.07	+0.05	+0.07	+0.08	+0.05	+0.05
E, max	−0.03	−0.05	−0.09	−0.04	−0.10	−0.10	−0.04	−0.05

900°, the values of the coefficients, c and d, being given only for comparison with their values at 1000° and 1100°.

The relatively low values of the correlation coefficients may be attributed to the small range in the observed values of true specific gravity, considering the experimental errors involved in its determination. The values of a are almost equal to the average values of V_K, and even the effect of ash content is small, owing primarily to the fact that the specific gravity of the ash is not greatly different from that of the coke. The probable errors of the calculated values are not greater than the probable errors of the experimental determination. Deferring discussion of the following statement to a later paper, it may be said that the so-called "true" specific gravities of cokes determined by the American Society for Testing Materials method used here are unquestionably too low.

Shatter Test.—The shatter-test data as reported for the cokes considered herein give the percentage of the coke sample—subjected to the A.S.T.M. test but using a 25-lb. rather than a 50-lb. sample—which is retained on 2-in., 1.5-in., 1-in., and 0.25-in. screens. For a shatter "index," the percentage retained on the 1.5-in. screen is used. This procedure is unsatisfactory, for at least two reasons: (1) particularly for the cokes prepared at lower temperatures, the index may be determined by less than 5 per cent of the sample—the test is not *severe* enough; and (2) the use of any single screen size as an index takes a single point from a curve describing the size distribution whereas any size distribution requires a minimum of two constants for its description.[28] In the present instance nothing can be done to overcome the first objection, but analysis of the available data has shown that the second objection may be at least partly nullified. The method followed is outlined below.

In considering the analysis of shatter-test data, it has seemed best to first present a purely hypothetical case and then to show how far this case may be applicable to the data on the assay cokes. In Fig. 3 are plotted, for two cokes having the same

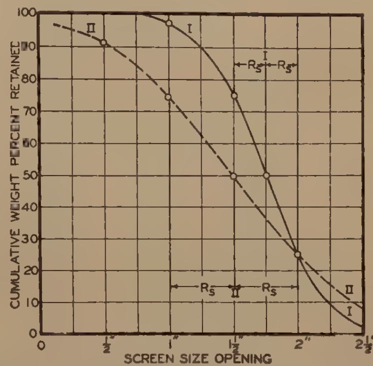

FIG. 3.—DIRECT PLOT OF HYPOTHETICAL SHATTER TEST DATA FOR TWO COKES, I AND II.
Curves as drawn are probability integrals. Fifty per cent of the sample is within $\pm R_s$ of the mean size.

2-in. shatter index, the cumulative weight per cent retained on 0.5-in., 1-in., 1.5-in., and 2-in. screens. The curves as drawn are the probability integrals: for coke I, the curve is completely described by saying it is that for a sample having an average size of 1.75 in. and that 50 per cent of the material falls within the limits 1.75 ± 0.25 in. with a normal probability distribution of sizes; and, for Coke II, similarly 1.5 ± 0.5 in. In general, shatter-test data on cokes, when plotted as in Fig. 3, do not permit accurate determination of the constants of the curve, but when plotted on arithmetic probability paper as in Fig. 4, the same data yield straight lines from which the constants can be determined easily. The average size, M_s, is that at 50 per cent cumulative weight per cent retained and the spread, R_s, is the difference in size between the 50 per cent value and that at either 25 or 75 per cent. It has been found that shatter-test data in general, not only for the assay cokes, give straight lines when plotted as in Fig. 4, except that often the amount of minus 0.5 or 0.25-in., is a few per cent greater than would be indicated

by the plus 1-in. material. This may be attributed to attrition and not to shatter.

While cokes I and II have the same 2-in. shatter index, their average sizes differ and, because of the different "spreads," 96 per cent of coke I lies between 1 and 2.5 in., while only 67 per cent of coke II lies between the same limits. This indicates that no single index is adequate for expressing shatter-test data, but that both an average size, M_s, and a spread value, R_s, are required, which may be obtained directly from the data.

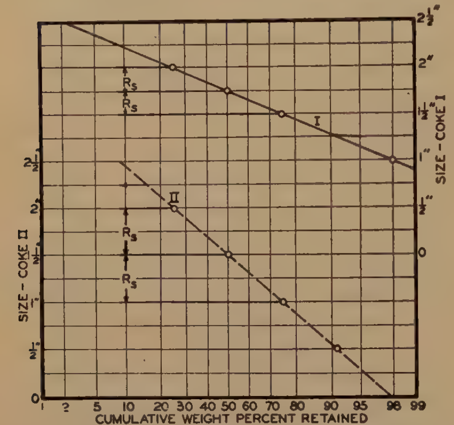

FIG. 4.—PLOT OF THE DATA OF FIGURE 3 ON ARITHMETIC PROBABILITY COORDINATES.

Departing from the general to the specific case, there are plotted in Fig. 5 the shatter-test data obtained for the 1000° assay cokes, 13-in. retort, for coal No. 7, unwashed and containing 15.9 per cent ash, and for coal No. 8, from the same seam but washed and containing only 8.3 per cent ash. A first glance at the lines drawn may lead one to conclude that they do not closely fit the data; this is due to inequality of the scale and closer inspection will show that no point is more than 2 per cent away from the line drawn. The coke from the unwashed coal has an average size, M_s, of 2.00 in. and a spread, R_s, of ± 0.55 in., while the values for the coke prepared from the washed coal are 1.62 ± 0.32 in. It may be noted that the 2-in. shatter of the "unwashed" is almost double that of the "washed" coke and also that "unwashed" coke has a greater average size, yet, owing to the lower R_s value of the "washed" coke, it has 86 per cent between the 1-in. and 2.5-in. limits, while the "unwashed" coke has only 63 per cent between the same limits. The increased *uniformity* in size of the "washed" coke is measured directly by the decreased R_s value.

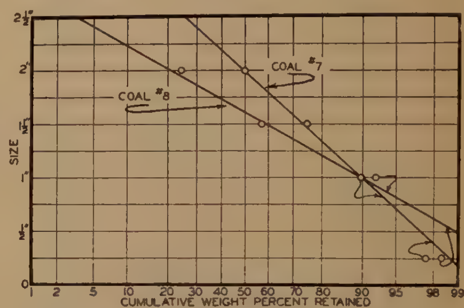

FIG. 5.—PLOT OF SHATTER-TEST DATA FROM COKES MADE AT 1000°C. FROM COALS NOS. 7 AND 8.

Coal No. 7, Mary Lee seam, unwashed, 15.9 per cent ash.

Coal No. 8, Mary Lee seam, washed, 8.3 per cent ash.

This study of the significance of shatter-test data on cokes, only partly outlined here, directed the method to be followed in analysis of the data for the assay cokes. The data were all plotted on arithmetic probability paper and values of M_s and R_s were determined. These values were then related for each carbonization temperature to the coal analyses, with the results shown in Table 10. It was necessary in this case to treat the data on cokes prepared in the 13-in. retort separately from the data of the 18-in. retort. The equations developed and the errors of the calculated values are given in Table 10. The equations for M_s and R_s are:

$$M_s = a + bFC^* + cA_C + d(FC^* - 65)^2 \qquad [20]$$

$$R_s = a + bFC^* + cA_C \qquad [21]$$

Included also in Table 10 are the 1.5-in.

TABLE 10.—*Shatter Test on Cokes*

$$M_s = a + bFC^* + cAc + d(FC^* - 65)^2 \quad [20]$$

T.,C.	500°	600°	700°	800°		900°		1000°		1100°
Retort diameter, in.	13	13	13	13	18	13	18	13	18	13
n	42	49	50	45	23	48	73	28	39	22
M_s avg.	3.86	3.09	3.09	2.57	2.77	1.85	1.93	1.50	1.58	1.30
M_s max.	7.50	8.45	8.90	4.15	4.30	2.80	2.55	2.00	1.90	1.65
M_s min.	1.15	1.65	1.65	1.40	1.90	1.30	1.25	1.10	1.30	1.00
R	0.52	0.74	0.78	0.73	0.92	0.78	0.71	0.79	0.80	0.85
a	−3.644	−6.848	−7.206	−2.087	−1.768	−0.712	0.336	−0.174	0.672	0.109
b	0.1145	0.1511	0.1560	0.0680	0.0632	0.0353	0.0214	0.0234	0.0123	0.0162
c	0	0	0	0.0452	0.0546	0.0552	0.0341	0.0324	0.0181	0.0276
d	0	0	0	−0.00231	0	−0.00183	0	−0.00135	0	−0.00103
$PE_{[20]}$	0.85	0.67	0.63	0.24	0.17	0.12	0.09	0.08	0.05	0.05
E, max.	+4.23	+2.52	+1.67	+0.93	+0.44	+0.38	+0.50	+0.27	+0.21	+0.14
E, max.	−3.17	−4.78	−5.25	−0.92	−0.56	−0.53	−0.54	−0.32	−0.18	−0.17
$R_s = a + bFC^* + cAc$ [21]										
R_s avg.	1.29	0.95	0.95	0.69	0.76	0.42	0.43	0.35	0.36	0.30
R_s max.	3.25	2.55	2.75	1.20	1.25	0.85	0.70	0.55	0.50	0.45
R_s min.	0.60	0.55	0.50	0.45	0.55	0.30	0.35	0.25	0.30	0.15
R	0.59	0.67	0.68	0.48	0.75	0.74	0.51	0.62	0.60	0.52
a	−1.659	−1.609	−1.896	0.382	−0.116	0.249	0.316	0.247	0.271	0.212
b	0.0450	0.0390	0.0432	0.0025	0.0108	0	0	0	0	0
c	0	0	0	0.0250	0.0295	0.0303	0.0229	0.0168	0.0171	0.0130
$PE_{[21]}$	0.28	0.21	0.24	0.088	0.069	0.051	0.041	0.045	0.029	0.045
E, max.	+1.14	+0.75	+0.60	+0.28	+0.17	+0.12	+0.12	+0.09	+0.07	+0.14
E, max.	−1.14	−1.44	−1.64	−0.35	−0.31	−0.22	−0.22	−0.17	−0.09	−0.14
1.5-INCH SHATTER INDEX, I, CALCULATED VALUES FROM M_s AND R_s										
Avg. I, obs., per cent	86.2	81.2	80.5	84.4	85.9	69.5	75.0	48.3	54.4	31.3
Max. I, obs., per cent	96.5	96.9	97.4	95.5	96.1	89.5	89.4	74.2	75.1	56.9
Min. I, obs., per cent	38.9	50.2	46.8	43.8	66.5	36.1	52.6	18.6	39.5	15.3
PE, cal., per cent	5.12	4.60	4.42	3.63	2.76	4.66	3.60	6.74	4.70	5.23
E, max., per cent	+45.7	+30.6	+32.3	+23.7	+11.3	+13.7	+31.7	+20.8	+22.5	+12.2
E, max., per cent	−6.5	−33.4	−32.7	−11.8	−6.0	−22.5	−14.5	−22.3	−13.9	−13.9

shatter indices calculated from the values of M_s and R_s obtained from these equations. The 2-in. shatter index, or any other specific size, may also be determined from Eqs. 20 and 21, either graphically or by use of probability integral tables. It may be said that when M_s and R_s are correlated with ash, and with carbon and hydrogen from the ultimate analyses, the high-temperature values are more closely related to the ultimate analysis than to the proximate analysis, while the contrary is true for low-temperature cokes.

Several points of interest result from consideration of the data in Table 10, some of which may be mentioned here. The ash content of the parent coal has no significant effect on either the average size or the "spread" at carbonization temperatures below 800°, except for the cokes prepared at 800° in the 18-in. retort. In general, the shatter-test data obtained on cokes prepared in the 18-in. retort closely approximate those obtained for cokes prepared in the 13-in. retort at a lower temperature. At 900° and above, the "spread" is not significantly dependent on the dry, ash-free fixed carbon of the parent coal, but is determined solely by the ash content of the coal. The inclusion of a second-order term in fixed carbon $(FC^* - 65)^2$, at the higher temperatures, simply indicates that the relation of M_s to FC^* is not linear over the entire temperature range.

Previous analysis of shatter-test data from other sources indicated that the ash-softening temperature was a significant factor. The statistical analysis showed that there was a small but definitely significant correlation (about 0.21) between M_s and ash-softening temperature; that is, resistance to breakage by shatter increases slightly with ash-softening temperature. There was no significant correlation of R_s with ash-softening temperature.

Again, it must be emphasized that the numerical coefficients in Eqs. 20 and 21 cannot be applied to data obtained in carbonization equipment other than the cylindrical retorts used in obtaining the data considered herein. The data are not sufficiently regular to warrant an attempt to determine the temperature dependence of the coefficients.

Tumbler Test.—Analysis of tumbler-test data indicates that the test itself may measure a combination of properties of coke such as might be imagined on consideration of the method of performing the test; a part of the degradation in size being due essentially to breaking by shatter and a part to abrasion. None of the usual methods of plotting size distributions[28] gives a straight line for the cokes after the tumbler test. The data can be reproduced by two distributions, each of which is a straight line on arithmetic probability paper, but the data available are inadequate to determine these distributions. There is a relatively high correlation between the percentage of the sample retained on the 1-in. or larger screen and the shatter-test data, which suggests that the degradation to 1 in. is largely the result of shatter. The correlation between the percentage on, or through, the 0.25-in. screen and shatter indices is not high, and so this figure may be assumed to measure approximately another property of coke; namely, the resistance to abrasion or attrition.

There is no evidence from the data presented that the percentage retained on the 0.25-in. screen in the tumbler test, B_K, for cokes prepared from a given coal at a fixed carbonization temperature is dependent on the retort diameter. This is a significant difference from the shatter-test data. Where values were given for cokes prepared in both the 13-in. and the 18-in. retorts, an average was taken and considered a single observation. The equation developed to relate B_K to coal analysis is:

$$B_K = a + bFC^* + cA_C + d(FC^* - 65)^2 \qquad [22]$$

The symbols have the same significance

assigned to them heretofore. The probable errors shown in Table 11 are not much greater than the error of the experimental determination, though the maximum positive and negative errors are considerably higher than might be expected from accidental errors or lack of reproducibility of the test data. A study of the coals giving the maximum errors does not help to clarify the situation: for instance, cokes prepared from coals 10, 19 and 32 give at different temperatures extreme errors, but the extreme errors are not always of the same sign, being at one temperature positive and at another negative in sign.

In this case also the numerical values of the coefficients must be limited to data obtained in the BM-AGA assay test. Their dependence on temperature is not regular, though it may be observed that the average value of B_K increases regularly with carbonization temperature, indicating that the resistance of cokes to abrasion increases with the temperature of preparation. Also, the correlation coefficient has a minimum value for the 700° cokes. Not only in this respect, but in others, the assay data obtained at 700° appear more difficult to predict from coal analysis than similar data obtained at either lower or higher temperatures. Many previous studies have indicated that a temperature range close to 700° is "critical"[19] and the present data would suggest that this critical range is slightly below 700° for some coals and slightly above for others. This point would seem to warrant much further study.

TABLE 11.—*Percentage Retained on 0.25-inch Screen in Tumbler Test, B_K*

$$B_K = a + bFC^* + cAc + d(FC^* - 65)^2 \quad [22]$$

$T.$,C.	500°	600°	700°	800°	900°	1000°	1100°
n	42	51	50	53	87	52	22
B_K avg., per cent	57.2	60.9	65.7	69.2	73.0	75.3	76.7
B_K max., per cent	70.5	72.9	75.1	76.8	80.1	82.0	84.9
B_K min., per cent	21.6	34.3	47.6	56.8	59.0	62.0	70.1
R	0.58	0.62	0.43	0.62	0.59	0.54	0.66
a	−0.55	27.50	51.79	44.04	64.01	62.63	58.41
b	0.999	0.629	0.274	0.443	0.201	0.239	0.323
c	−0.739	−0.777	−0.584	−0.444	−0.804	−0.550	−0.445
d	−0.080	−0.062	−0.015	−0.029	0	0	0
$PE_{[22]}$ per cent	5.0	3.7	3.0	2.1	2.0	2.4	2.0
E, max., per cent	+24.1	+21.9	+20.5	+6.5	+9.5	+9.1	+7.2
E, max., per cent	−16.6	−7.8	−8.7	−10.7	−9.5	−7.1	−4.1

CONDENSABLE BY-PRODUCTS

When considering by-product yields, a factor that can be neglected for coke must be given some weight. This factor is the material balance in the assay test. Since the retort is placed in a hot furnace and connections are then made, opportunity is given for loss of volatile products and the data show, as might be expected, that this loss tends to increase with carbonizing temperature. For this reason, data from assay tests in which the material balance was less than 98 per cent were not included in determining the dependence of by-product yields on coal analysis. From internal evidence, the data show that manipulative errors of this kind decreased as experience with the assay test increased.

Tar plus Light Oil

The distinction between tar and light oil is based largely on volatility and the distribution is at least partly dependent on the efficiency of scrubbers and cooling towers. For this reason the first correlation studies were made on the sum of the two constituents.

A comparison of the yields of tar plus light oil, $(T + LO)$, from the tests in the 13-in. and in the 18-in. retorts showed that on the average the yield from the 13-in. retort was 0.4 per cent greater than from the 18-in. retort. For further analysis of the data, where tests were made in both retorts, 0.4 per cent was added to the yield obtained in the assay using the 18-in. retort and an average of this value and that directly recorded for the 13-in. retort was taken as a single observation. This procedure was supported by separate analysis of the data obtained from the two sizes of retort.

When considering high-rank bituminous coals only, the yield of tar plus light oil might be expected to be directly proportional to the volatile-matter content of the coal. However, lower-rank coals may have high volatile-matter content and yield little tar plus light oil. Fortunately for the type of analysis attempted here, these lower-rank coals also have a greater natural moisture content than do the higher-rank coals. As shown in Table 12, inclusion of terms both in volatile-matter content and moisture content gives an equation that reproduces satisfactorily the observed yields of tar plus light oil; i.e.,

$$(T + LO) = a + b(VM) + c(H_2O) \quad [23]$$

It should be emphasized that the moisture content is not considered as having any direct effect on the $(T + LO)$ but is used in Eq. 23 solely as an index of rank of the coal.

The correlation coefficients are very high and the errors of the calculated values are about of the magnitude expected for the experimental determination (Table 12). When this equation was applied to the data previously excluded because of low recovery, it was found that the errors of calculated values were generally positive, indicating that the low recovery is at least partly due to loss of tar and light oil. Hence, the omission of these values from consideration in determining the constants of the equation is justified.

TABLE 12.—*Yield of Tar plus Light Oil, $(T + LO)$*

$(T + LO) = a + b(VM) + c(H_2O)$ [23]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	45	48	45	50	83	46	17
$(T + LO)$, avg., per cent	8.2	8.4	7.6	7.2	6.7	6.1	5.4
$(T + LO)$, max., per cent	12.7	11.8	12.0	11.0	9.3	8.9	7.8
$(T + LO)$, min., per cent	1.6	1.8	1.9	2.2	1.6	1.7	1.1
R	0.96	0.96	0.96	0.96	0.95	0.97	0.95
a'	−5.25	−4.17	−3.21	−2.35	−2.01	−1.94	−2.45
b'	0.447	0.427	0.375	0.331	0.304	0.281	0.267
c'	−0.158	−0.202	−0.223	−0.211	−0.208	−0.199	−0.204
$PE_{[23]}$, per cent	0.61	0.59	0.50	0.42	0.35	0.33	0.35
E, max., per cent	+2.1	+1.6	+1.6	+1.4	+1.4	+1.1	+1.1
E, max., per cent	−2.1	−1.7	−1.6	−1.4	−1.1	−1.2	−0.7
a		−4.10	−3.09	−2.41	−2.06	−2.04	−2.35
b		0.427	0.373	0.334	0.305	0.282	0.264
c		−0.218	−0.214	−0.210	−0.206	−0.202	−0.198

The errors given in Table 12 are those calculated from the equation using the values a', b' and c', which were determined by least squares for the data obtained at the individual temperatures. Except for the values of these coefficients at 500°, they are related to temperature by the following equations:

$$\log b = \frac{498.7}{T + 273} - 0.9411 \quad [24]$$

$$c = 0.00004T - 0.242 \quad [25]$$

$$a = -17.13 + 0.03166T - 0.1656 \times 10^{-4}T^2 \quad [26]$$

The values of the coefficients calculated from these equations are given in Table 12 as a, b and c for each temperature from 600°

to 1100°. Their use for calculating yields of $(T + LO)$ is to be preferred over the use of a', b' and c'. At 500°, the constants calculated from Eqs. 24, 25 and 26 would give yields of $(T + LO)$ averaging 1.4 per cent higher than the observed, which is in accord with the analysis of the data on carbonizing time that showed that in order to make the data consistent with the observed carbonizing times at the higher temperatures it would have been necessary to extend the average carbonizing time at 500° approximately 10 hr. Except at 500°, the use of a, b and c gives almost exactly the same yields of $(T + LO)$ as does the use of the original least-squares coefficients; rarely is the calculated yield changed by as much as 0.1 per cent. It should be noted that the Eqs. 24, 25 and 26 apply to the data obtained from both the 13-in. and the 18-in. retorts except that the first term on the right-hand side of Eq. 26 must be changed to -17.53 in calculating the yield of $(T + LO)$ from the 18-in. retort.

Light Oil

The yields of light oil, (LO), were also correlated separately with the volatile-matter content of the coal. The equation is given by the expression:

$$(LO) = a + b(VM) \qquad [27]$$

In determining the coefficients, the data for the first 10 coals were not used, since a less efficient scrubber was used for the recovery of light oil in these cases[1]; also, the data were not used for experiments where the material balance was less than 98 per cent. The values of a' and b' given in Table 13 are those determined by least squares. The b' values are closely reproduced by the equation:

$$\log b = -0.5844 - \frac{1100}{T + 273} \qquad [28]$$

and the values of b calculated from this equation are also given in Table 13 and should be used in preference to the b' values. The values of a' at each temperature were then adjusted to give zero average error, using the values of b, and these adjusted values are shown as a in the table. The errors in the calculated values of light oil are not significantly affected by the use of a and b instead of a' and b'.

Ammonium Sulphate

The yield of ammonium sulphate at the different temperatures showed only a slight correlation with the nitrogen content of the coal as carbonized, N_C. However, there is much closer correlation in the yield with the volatile nitrogen of the parent coal, N_V. When coal is heated, a part of the nitrogen of coal is retained in the coke, which is equal to the percentage of nitrogen in the coke, N_K, multiplied by the yield of coke, $K/100$. The values of N_K can be calculated by an equation of the form of Eq. 9; i.e.,

$$N_K = N_C + D_U \qquad [29]$$

where the values of D_U are those given in Table 6. The yield of coke is given by Eq. 4. Then

$$N_V = N_C - (N_C + D_U)K/100 \qquad [30]$$

The volatile nitrogen will be distributed in the carbonization products, in part as ammonia, in part as organic compounds in the tar and gas, and in part as free nitrogen in the gas. As is well known, the presence of moisture in the coal helps to prevent thermal decomposition of ammonia and for this reason, in part, a term in moisture should be used in calculating the yield of ammonia. The inclusion of the term in moisture may also be justified (1) as an index of rank, because in general the nitrogen in low-rank coals is commonly believed to be in functional groups more apt to yield ammonia on heating than that in higher-rank coals, and (2) as a recognition of the fact that a direct action of steam on coke nitrogen is known to yield ammonia. It would be impossible to separate these three

functions of moisture in the type of analysis reported here.

An average of the yields of ammonium sulphate, *Am*, in pounds per ton from the 13-in. and the 18-in. retorts was used where both were given, since the average difference in yield from the two retorts was only 0.1 lb. per ton. The equation used to obtain the results shown in Table 14 is:

$$Am = a + bN_V + c(H_2O) \qquad [31]$$

The yields so calculated are consistently high for coals 8 and 9 and consistently low for coals 24 and 26. The probable errors of the calculated values are not much greater than the probable error of the observed values as calculated from the differences between the yields from the 13-in. and 18-in. retorts, which is ±0.96 lb. per ton.

The use of the calculated coke nitrogen (Eq. 30), particularly at 500°, was found to be superior to the use of the directly determined coke nitrogen, and this may be attributed to averaging out of errors due to inadequacy of existing analytical methods for determining nitrogen in coke.

TABLE 13.—*Yield of Light Oil*

$(LO) = a + b(VM)$ [27]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	40	42	41	44	78	43	19
(LO) avg., per cent	0.31	0.46	0.63	0.75	0.95	0.94	0.93
(LO) max., per cent	0.51	0.73	1.00	1.10	1.33	1.40	1.24
(LO) min., per cent	0.11	0.23	0.25	0.40	0.45	0.32	0.28
R	0.75	0.58	0.85	0.82	0.84	0.93	0.91
a'	−0.020	0.093	−0.034	0.044	−0.049	−0.214	−0.338
b'	0.0107	0.0120	0.0213	0.0226	0.0325	0.0374	0.0390
$PE_{[27]}$, per cent	0.046	0.082	0.063	0.076	0.076	0.074	0.073
E, max., per cent	+0.19	+0.36	+0.18	+0.34	+0.27	+0.29	+0.20
E, max., per cent	−0.12	−0.22	−0.21	−0.24	−0.30	−0.26	−0.16
a	0.008	0.022	0.029	−0.018	0.028	−0.158	−0.410
b	0.0098	0.0143	0.0193	0.0246	0.0300	0.0356	0.0412

TABLE 14.—*Yield of Ammonium Sulphate, Am*

$Am = a + bN_V + c(H_2O)$ [31]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	50	52	51	54	88	53	30
Am, avg., lb. per ton	4.3	10.8	21.7	22.0	19.5	14.4	11.7
Am, max., lb. per ton	8.4	16.9	31.5	33.1	24.8	19.4	16.9
Am, min., lb. per ton	1.1	4.8	11.1	14.6	12.1	8.1	5.1
R	0.69	0.82	0.69	0.70	0.64	0.76	0.47
a	2.51	4.84	10.44	12.86	9.71	−4.69	−17.33
b	10.3	18.8	23.6	17.4	18.7	29.6	32.1
c	0.106	0.270	0.398	0.447	0.212	0.024	−0.128
$PE_{[31]}$, lb. per ton	0.74	1.1	2.1	1.7	1.4	1.4	2.1
E, max., lb. per ton	+1.9	+3.8	+7.0	+5.2	+5.6	+5.1	+5.6
E, max., lb. per ton	−4.4	−3.7	−9.1	−7.3	−6.0	−4.4	−7.4

In this case again, the numerical values of the coefficients should not be used except for BM-AGA assay data. As the investigators themselves point out, the observed yields are particularly low at high temperatures in comparison with coke-oven experience, and this is attributed to cracking of the ammonia on the iron surfaces of the retorts.[13] The dependence on temperature of the values obtained for the coefficients was not sufficiently regular to justify an attempt to find equations for expressing it. In general, however, the increase in the value of b with temperature is to be expected on the basis of decomposition of the more complex organic compounds to ammonia. The reason for the decrease in the value of c above 800° is not obvious.

Liquor

The aqueous liquor obtained in coal carbonization is derived in part from the moisture in the coal and in part from thermal decomposition of the organic matter of the coal. As an index of the chemical nature of the organic matter, the volatile matter on a dry, ash-free basis, VM^*, is used in the following equation, developed for calculating the percentage by weight of liquor obtained in the assay tests, L:

$$L = a + bVM^* + c(H_2O) \qquad [32]$$

TABLE 15.—*Yield of Liquor, L*

$L = a + bVM^* + c(H_2O)$ [32]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	50	52	51	54	88	53	30
L, avg., per cent	6.4	7.1	7.2	6.8	6.3	5.8	4.6
L, max., per cent	30.3	29.1	26.8	24.6	21.3	21.0	13.9
L, min., per cent	1.3	1.7	2.3	2.1	1.5	1.4	0.9
R	0.99	0.99	0.99	0.99	0.97	0.98	0.92
a	−2.17	−1.81	−1.49	−1.40	−1.55	−1.75	−1.85
b	0.140	0.160	0.158	0.157	0.157	0.150	0.123
c	1.298	1.225	1.121	1.017	0.965	0.858	0.875
$PE_{[32]}$, per cent	0.50	0.46	0.47	0.40	0.52	0.42	0.63
E, max., per cent	+1.6	+1.3	+1.7	+1.3	+2.3	+1.3	+1.9
E, max., per cent	−2.2	−2.5	−1.6	−1.3	−3.0	−1.4	−1.6

The correlation coefficients are very high, as may be seen from the data of Table 15, which also gives the errors of the calculated values. In this analysis, the values obtained from the 13-in. and 18-in. retorts were averaged and used as a single observation. It seems likely that the probable errors are largely accounted for by errors in the experimental determination. The fact that the coefficient c decreases with increasing carbonization temperature may probably be attributed largely to the increase of water-gas formation with temperature; this is supported by an analysis of the distribution of oxygen in the carbonization products not included in this discussion. The following single equation,

$$L = -10.71 + 0.02416T - (0.1532 \times 10^{-4})T^2 + 0.152VM^* + (1.722 - 0.852 \times 10^{-3}T)(H_2O) \qquad [33]$$

includes the temperature dependence of the coefficients and reproduces the whole set of 378 observations with a probable error of ±0.49 per cent.

Gas

Yield as Weight Per Cent

The method followed in developing an equation for calculating the per cent weight of gas obtained on carbonization G_W was identical with that followed for the yield of tar plus light oil, except that the data for coal No. 49 were omitted in determining the numerical coefficients and that the data for the two sizes of retorts were treated separately. The equation used, therefore, in calculating the results shown in Table 16 is:

$$G_W = a + b(VM) + c(H_2O) \qquad [34]$$

The term in moisture has no significant effect on the results obtained in the 13-in. retort at the three lowest temperatures and is of doubtful significance at the two highest temperatures. The effect of moisture is much more important in the 18-in. retort. From a consideration of the magnitude of the errors of the calculated values, it appears that Eq. 34 fits the data within approximately the error of the experimental determination.

A study was made of the dependence on temperature of the coefficients a', b' and c', which were determined by least-squares

methods for each temperature. This study was limited to the data obtained in the tests using the 13-in. retort. The values of a' were fitted to the equation:

$$a = 7.34 - \frac{408}{T - 403} \qquad [35]$$

This yields the calculated values of a shown in Table 16. The values of b' determined

coefficients are in conformity with the statistical tests of significance of the effect of moisture at 1000° and 1100° in comparison with the effect at 800° and 900°. As in other similar cases, previously discussed, the use of the values of a, b and c is to be preferred to the individual least-squares values a', b' and c'. With data obtained at only three temperatures in the 18-in. retort, a similar analysis is not justified.

TABLE 16.—*Yield of Gas, Per Cent by Weight, G_W*

$$G_W = a + b(VM) + c(H_2O) \quad [34]$$

T., C.	500°	600°	700°	800°		900°		1000°		1100°
Retort diameter, in.	13	13	13	13	18	13	18	13	18	13
n	45	48	45	41	22	43	73	24	37	18
G_w, avg., per cent	5.8	8.6	11.7	13.3	13.3	14.4	14.5	15.8	15.8	17.1
G_w, max., per cent	7.1	10.4	14.7	17.9	16.5	19.0	19.1	21.1	21.2	23.6
G_w, min., per cent	4.2	6.5	9.0	10.0	10.4	10.9	11.0	11.8	12.1	12.6
R	0.85	0.84	0.91	0.93	0.96	0.94	0.92	0.88	0.93	0.87
a'	3.21	5.13	6.18	6.31	6.41	6.52	6.48	6.61	7.41	6.73
b'	0.084	0.113	0.182	0.226	0.223	0.250	0.250	0.283	0.247	0.315
c'	0	0	0	0.090	0.117	0.115	0.154	0.085	0.323	0.153
$PE_{[34]}$, per cent	0.24	0.33	0.39	0.45	0.36	0.46	0.41	0.63	0.54	0.84
E, max., per cent	+0.7	+1.7	+1.1	+1.3	+1.3	+1.0	+1.5	+1.3	+1.7	+1.7
E, max., per cent	−1.0	−1.0	−1.5	−2.4	−0.9	−2.3	−2.2	−3.1	−2.7	−3.9
a	3.15	5.28	5.97	6.32	[a]	6.52	[a]	6.66	[a]	6.76
b	0.091	0.130	0.169	0.208	[a]	0.247	[a]	0.286	[a]	0.326
c	0	0	0	0.302	[a]	0.149	[a]	0.020	[a]	0.018

[a] See text.

TABLE 17.—*Yield of Gas as Percentage of British Thermal Units in Coal Charge, G_H*

$$G_H = a + b(VM) + c(H_2O) \quad [37]$$

T., C.	500°	600°	700°	800°		900°		1000°		1100°
Retort diameter, in.	13	13	13	13	18	13	18	13	18	13
n	45	48	45	41	22	43	73	24	37	18
G_H, avg., per cent	8.0	12.4	17.3	19.9	20.4	21.5	21.9	22.9	23.2	23.9
G_H, max., per cent	9.8	13.7	20.0	23.6	23.0	25.0	25.6	27.2	27.8	29.4
G_H, min., per cent	6.1	10.6	13.4	17.0	17.5	17.6	17.3	18.6	18.8	19.2
R	0.70	0.68	0.83	0.88	0.97	0.91	0.93	0.88	0.95	0.86
a'	5.63	9.64	12.18	14.07	14.03	14.67	14.66	14.94	15.59	15.22
b'	0.077	0.089	0.166	0.185	0.201	0.213	0.225	0.251	0.224	0.265
c'	0	0	0	0.136	0.168	0.143	0.162	−0.003	0.288	0.113
$PE_{[37]}$, per cent	0.37	0.43	0.52	0.45	0.28	0.49	0.35	0.54	0.40	0.72
E, max., per cent	+1.1	+1.6	+1.9	+1.5	+0.9	+1.4	+1.1	+1.2	+0.8	+1.4
E, max., per cent	−1.5	−1.6	−1.4	−1.7	−1.0	−1.6	−1.5	−2.5	−2.2	−3.4
a	5.56	10.00	12.22	13.55	[a]	14.44	[a]	15.07	[a]	15.54
b	0.074	0.108	0.142	0.176	[a]	0.211	[a]	0.245	[a]	0.279
c	0	0	0	0.452	[a]	0.253	[a]	0.024	[a]	−0.213

[a] See text.

for 500°, 600°, and 700° were then adjusted to the calculated values of a and the following linear relation was found:

$$b = -0.106 + 0.000392T \qquad [36]$$

From these calculated values of a and b the values of c shown in the table were then determined. The adjusted values of the c

The only coal besides coal No. 49 that consistently gives extreme errors for the calculated values is one of the other lower-rank coals, Lower Sunnyside coal No. 19, and it, like No. 49, gives higher yields of gas than calculated. Coal No. 52, from the Pittsburgh seam, generally gives calculated yields lower than observed, although the

other coals from this seam do not show any marked trend in the same direction.

Yield of Gas as Percentage of British Thermal Units in Coal Charge

A second method of expressing the yield of gas obtained on carbonization is by using the B.t.u. in the gas as a percentage of the B.t.u. in the coal charged, G_H. The treatment of the data in this case is exactly, without exception, similar to that described in the preceding section. The equations are:

$$G_H = a + b(VM) + c(\mathrm{H_2O}) \qquad [37]$$

$$a = 18.85 - \frac{2644}{T - 301} \qquad [38]$$

and

$$b = -0.096 + 0.000341T \qquad [39]$$

The results of the calculations are summarized in Table 17. By this method of expressing yield, coal No. 19, like coal No. 49, generally gives observed values greater than the calculated ones. Coal No. 49 itself and coal No. 52 appear less extreme in their behaviors.

Specific Gravity of Gas

The yield of gas on coal carbonization is frequently reported in units of cubic feet per ton, and this volume is probably most conveniently measured in assay tests. The weight per cent is really a derived quantity from the volume and the specific gravity. However, in this study, it seemed advisable to correlate the weight per cent with analysis for direct comparison with the yields of other products. Knowing the weight per cent yield of gas and the specific gravity, the volume can be readily calculated. Furthermore, gas works are commonly under contract to supply gas of fixed specific gravity and this and the previously mentioned reasons led to the choice of a study of the correlation of specific gravity of the gas, rather than total volume, with analysis. It was found that the specific gravity of the gas, G_R, like its yield, is determined primarily by the volatile matter and moisture content of the parent coal as expressed in the equation:

$$G_R = a + b(VM) + c(\mathrm{H_2O}) \qquad [40]$$

TABLE 18.—*Specific Gravity of Gas,* G_R

$G_R = a + b(VM) + c(\mathrm{H_2O})$ [40]

T., C.	500°	600°	700°	800°	900°	1000°	1100°
n	50	52	51	54	88	53	29
G_R, avg.	0.606	0.523	0.443	0.395	0.365	0.354	0.344
G_R, max.	0.716	0.620	0.542	0.526	0.463	0.443	0.417
G_R, min.	0.506	0.433	0.351	0.300	0.269	0.255	0.246
R	0.75	0.83	0.87	0.85	0.90	0.95	0.90
a'	0.4323	0.3453	0.2685	0.2023	0.1793	0.1693	0.1455
$100b'$	0.509	0.532	0.514	0.581	0.548	0.537	0.579
$100c'$	0.531	0.418	0.507	0.454	0.649	0.644	0.547
$PE_{[40]}$	0.024	0.018	0.015	0.018	0.012	0.010	0.012
E, max.	+0.112	+0.069	+0.053	+0.115	+0.085	+0.028	+0.038
E, max.	−0.071	−0.066	−0.052	−0.063	−0.049	−0.035	−0.024
a	0.4399	0.3295	0.2618	0.2173	0.1860	0.1633	0.1460
$100b$	0.516	0.525	0.534	0.543	0.552	0.560	0.570
$100c$	0.467	0.490	0.513	0.536	0.559	0.582	0.605

Comparison of the data on specific gravity of gas obtained from the 13-in. retort with that obtained from the 18-in. retort at 800°, 900° and 1000° showed no significant difference and the values, where both were available, were averaged and counted as a single observation. From the observed differences between the values of G_R for the two sizes of retorts it may be concluded that the probable error of the experimental determination is of the order of one-half the probable error of the calculated value as given in Table 18. The temperature relationships of the coefficients

are given by the following expressions:

$$\log a = -1.4524 + \frac{847.0}{T + 273} \qquad [41]$$

$$100b = 0.470 + 9.07 \times 10^{-5}T \qquad [42]$$

$$100c = 0.452 + 22.93 \times 10^{-5}T \qquad [43]$$

The calculated values of a, b and c as given in the table are to be preferred to the individual least-squares values a', b', and c'. It may be said that the unwashed Mary Lee seam coal No. 7, the Lower Sunnyside coal No. 19, and the Michel seam coal No. 11 all tend to give specific gravities of the gas much higher than calculated from Eq. 40, while the Indiana No. 4 seam coal No. 45 and the blend of coals designated as coal No. 30 tend to give specific gravities of the gas considerably lower than calculated.

Sulphur in Gas

The behavior of the sulphur in a coal on carbonization should be similar to the behavior of nitrogen except that the appearance of free sulphur in the products does not seem as likely as the occurrence of free nitrogen. The sulphur in the gas, S_{G_1}, as percentage of sulphur in H_2S by weight of the coal charged does not appear to be influenced significantly by the moisture content of the coal, but is linearly related to the volatile sulphur, S_V, as

$$S_{G_1} = a + bS_V \qquad [44]$$

where S_V is given by the equation

$$S_V = S_T - S_K(K/100) \qquad [45]$$

in which S_K is the percentage of sulphur in the coke as given by Eq. 11, K is the percentage yield of coke given by Eq. 4, and S_T is the total sulphur in the coal. Sulphur in the gas is generally reported as grains of H_2S per 100 cu. ft., S_{G_2}, but can readily be calculated in percentage by the equation:

$$S_{G_1} = S_{G_2} \times G \times 6.720 \times 10^{-8} \qquad [46]$$

where G is the cubic feet per ton and 6.720×10^{-8} is a conversion factor. As may be seen from Table 19, Eq. 44 reproduces the data remarkably well, the correlation coefficients ranging from 0.92 to 0.97 and the probable errors from ± 0.02 to ± 0.04 per cent. The data for the 13-in. and the 18-in. retorts were averaged. The extreme errors do not appear to be related to coal analysis. A single equation,

$$S_{G_1} = -0.0588 + 0.1081 \times 10^{-3}T + (0.5906 - 0.1685 \times 10^{-3}T)S_V \qquad [47]$$

reproduces the entire 424 observations, having a correlation coefficient of 0.95 and gives a probable error of ± 0.029 per cent. Eq. 47 assumes that the coefficients a and b of Eq. 44 are linear functions of temperature and is preferred for use to the individual least-squares equations.

In this analysis the value of S_{G_1} is implicitly affected by the proximate analysis

TABLE 19.—*Sulphur in Gas, as Percentage by Weight of Coal Charged, S_{G_1}*
(SULPHUR AS H_2S)
$S_{G_1} = a + bS_V$ [44]

T.,C.	500°	600°	700°	800°	900°	1000°	1100°
n	48	50	49	66	116	66	29
S_{G_1}, avg., per cent	0.177	0.178	0.202	0.207	0.210	0.220	0.200
S_{G_1}, max., per cent	0.730	1.266	1.065	0.990	1.090	1.040	0.480
S_{G_1}, min., per cent	0.008	0.011	0.018	0.024	0.037	0.024	0.027
R	0.92	0.97	0.97	0.97	0.94	0.97	0.94
a'	0.0232	−0.0399	0.0140	0.0380	0.0494	0.0406	0.0586
b'	0.4431	0.5772	0.4658	0.4407	0.4298	0.4393	0.3690
$PE_{[44]}$, per cent	0.038	0.029	0.026	0.024	0.027	0.023	0.020
E, max., per cent	+0.165	+0.161	+0.106	+0.079	+0.199	+0.108	+0.066
E, max., per cent	−0.162	−0.091	−0.089	−0.069	−0.110	−0.084	−0.061
a	−0.0048	0.0060	0.0168	0.0277	0.0385	0.0493	0.0601
b	0.5064	0.4895	0.4727	0.4558	0.4390	0.4222	0.4053

of the coal as well as by the sulphur content of the coal, since the factor K in Eq. 45 is determined by the sum of fixed carbon plus ash of the coal. The sulphur content of the gas expressed as grains of H_2S per 100 cu. ft., S_{G_2}, can be calculated from Eqs. 46 and 44 but can also be calculated directly from

$$S_{G_2}/100 = a + bS_V + cVM + d(H_2O) \quad [48]$$

by use of the coefficients, and with the errors shown in Table 20.

U. S. Steel Corporation Assay Test

For comparison with the results obtained in the BM-AGA tests, the Bureau of Mines also generally made tests for determining carbonization yields by the Fischer low-temperature assay at 500°[29] and by the U. S. Steel Corporation assay at 900°.[30] The same methods of analysis were applied to the data obtained in the latter test as were used in the BM-AGA test in so far as data were available. Table 21 shows that the correlation of the assay carbonization results with proximate analysis in the U. S. Steel Corporation assay tests is just as satisfactory as the results described in the preceding paragraphs. In general, the data for all the 85 coals tested were used in determining the coefficients. However, the data for coal No. 45 and for the two blends made with it were omitted in correlating coke yield, and the data on tar yield and light oil yield in gallons per ton for the first 10 coals were not used, since analysis of the data indicated the presence of some systematic difference between these coals and those tested later.

Summary and Conclusions

In the preceding sections of this report, it has been shown that the yields of carbonization products and many of their properties may be calculated with reasonably small errors from the proximate analyses of coals subjected to the BM-AGA assay test. This work of the Bureau of Mines has provided a veritable wealth of data regarding the carbonizing qualities of American coals. Many of the data presented in the survey of American coals have not been subjected to the methods of analysis reported here. For any one interested in any particular result of the carbonization assay test, it may be suggested on the basis of the results reported in this paper that a similar analysis of the data would be well repaid.

In most of the equations relating yields of carbonization products, and many of their properties, to the proximate analysis of the coal, the coefficients modifying the analytical factors are markedly dependent on temperature. This fact is of great importance in coal carbonization. In general, a change in the analysis of the coal charged may be counterbalanced by a change in carbonizing temperature to maintain con-

Table 20.—*Sulphur in Gas as Grains* H_2S *per* 100 *Cubic Feet*, S_{G_2}

$$S_{G_2}/100 = a + bS_V + cVM + d(H_2O) \quad [48]$$

T., C.	500°	600°	700°	800°	900°	1000°	1100°
n	48	50	49	52	84	51	29
$0.01S_{G_2}$, avg.	10.5	6.1	4.3	3.5	3.0	2.8	2.3
$0.01S_{G_2}$, max.	49.4	46.5	23.8	16.0	16.3	13.1	5.8
$0.01S_{G_2}$, min.	0.6	0.4	0.4	0.4	0.5	0.3	0.3
R	0.96	0.98	0.98	0.97	0.97	0.98	0.96
a	−4.98	−3.54	−0.90	0.70	0.99	0.64	0.64
b	27.14	20.80	10.27	7.33	7.06	5.78	5.05
c	0.131	0.091	0.051	0	0	0	0
d	0.833	−0.458	−0.233	−0.074	−0.0271	−0.092	−0.126
PE	1.68	0.92	0.45	0.38	0.32	0.28	0.20
E, max	+6.1	+4.1	+1.7	+1.2	+1.0	+1.0	+0.7
E, max	−7.6	−3.4	−1.4	−1.7	−1.5	−0.9	−0.6

TABLE 21.—*Results of U. S. Steel Corporation Assay Tests*

	Coke, Per Cent	Tar plus Light Oil, Per Cent	Light Oil, Per Cent	Gas, Per Cent	Liquor, Per Cent	$\frac{\text{B.t.u. Gas}}{\text{B.t.u. Coal}}$ Per Cent	Specific Gravity of Gas	Tar, Gal. per Ton	Light Oil, Gal. per Ton	Ammonium Sulfate, Lb. per Ton	Gas, Cu. Ft. per Ton
n	82	85	85	85	85	85	85	75	75	85	85
Average value	73.9	5.16	1.06	13.6	7.00	20.8	0.334	8.54	2.81	24.23	10,410
Maximum value	86.2	7.34	1.71	17.6	20.2	24.2	0.409	11.9	4.70	32.6	11,400
Minimum value	61.0	1.89	0.49	8.9	2.4	16.4	0.243	2.9	1.34	18.6	9,300
R	0.98	0.91	0.77	0.95	0.99	0.90	0.94	0.96	0.79	0.83	0.64
Constant term in equation	24.14	−1.23	−0.043	4.21	−0.97	13.6	0.1478	−2.34	−0.078	10.12	9,740
Coefficient of FC + ash	0.742										
Coefficient of VM		0.220	0.0357	0.306		0.235	0.00571	0.376	0.0941		30.8
Coefficient of (H_2O)		−0.148			1.121		0.00436	−0.273		0.404	−109
Coefficient of VM*					0.155						
Coefficient of N_v										27.96	
PE	0.54	0.35	0.11	0.36	0.32	0.40	0.0081	0.39	0.27	1.1	210
E, max	+3.0	+1.89	+0.71	+1.1	+1.1	+1.5	+0.028	+1.2	+1.81	+4.3	+740
E, max	−1.6	−0.99	−0.45	−2.0	−1.6	−1.9	−0.040	−1.3	−1.13	−4.5	−720

stant a specific result. Uniform yields and properties of products from charges of constant chemical analysis cannot be obtained without accurate control of the carbonizing temperature. Where variations both in coal analysis and in carbonizing temperature occur at random, considerable nonuniformity in yields and properties of the products will result; data are not available to permit an estimate of the importance of these random variations in determining the nonuniformity of results of commercial coke-oven operation.

Since the equations developed for expressing carbonization data as functions of coal analyses have been based on coals covering such a wide range of rank, it should be expected that they have general applicability to other coals. How far this may be true may be illustrated by the data in Table 22, which gives the results of application of the relations found to some of the data reported on coal No. 57 from the Pocahontas No. 4 seam and on two blends of this coal with a Pittsburgh seam coal. The calculated results obtained from coal No. 56 and its blends are equally satisfactory, as are also data not presented here but which may be calculated from the equations given. The data obtained in the assay tests of coals 56 and 57 were published[31] while the study described in the present paper was in progress and were not used in determining the coefficients of the equations. A test of the general applicability of the relations found based on coal No. 57 is rather severe, since the Pocahontas No. 4 seam coal is very near the extreme, on the high-rank side, of the coals tested. However, the agreement between calculated and observed carbonization yields will, it seems likely, be generally regarded as satisfactory.

It may be stated that the general relations given in this paper are most uncertain for lower-rank coals. These coals are the largest reserves of solid fuel in the United States and should receive further study.

More data on the lower-rank coals should make it possible to understand, or predict, better the effects of conditions of carbonization on the yields and properties of the data for blends appeared generally to be additive based on the original coals and that the extreme petrographic "types" of coal did not in general differ significantly

TABLE 22.—*Comparison of Observed and Calculated Carbonization Yields and Shatter and Tumbler Indices for a Pocahontas No. 4 Seam Coal (No. 57) and for Two Blends with a Pittsburgh Seam Coal (No. 57A, No. 57B)*

	Coal No. 57								Coal No. 57*A*	Coal No. 57*B*
T.,C.	500°	600°	700°	800°		900°		1000°	900°	900°
Retort diameter, in.	13	13	13	13	18	13	18	18	18	18
Coke, per cent:										
Observed	90.8	88.5	85.7	84.5	84.5	84.2	84.3	84.4	72.4	74.0
Calculated	92.1	88.1	85.6	84.6	84.6	84.5	84.5	84.4	72.1	73.4
E	1.3	−0.4	−0.1	0.1	0.1	0.3	0.2	0.0	−0.3	−0.6
Gas, per cent:										
Observed	4.4	6.3	9.0	10.1	10.4	10.6	10.6	11.3	14.1	13.6
Calculated	4.6	6.9	9.1	10.1	10.1	10.7	10.7	11.8	14.6	14.2
E	0.2	0.6	0.1	0.0	−0.3	0.1	0.1	0.5	0.5	0.6
Tar + light oil, per cent:										
Observed	2.1	2.4	2.3	2.6	2.4	2.1	2.1	2.0	7.0	6.35
Calculated	1.7	2.4	2.5	2.7	2.3	2.6	2.2	1.9	6.9	6.30
E	−0.4	0.0	0.2	0.1	−0.1	0.5	0.1	−0.1	−0.1	−0.05
Light oil, per cent:										
Observed	0.24	0.31	0.38	0.47	0.46	0.46	0.48	0.48	1.03	0.95
Calculated	0.16	0.25	0.34	0.38	0.38	0.51	0.51	0.41	0.98	0.93
E	−0.08	−0.06	−0.04	−0.09	−0.08	0.05	0.03	−0.07	−0.05	−0.02
Liquor, per cent:										
Observed	2.3	2.6	3.0	3.0	2.6	1.9	2.4	2.3	5.6	5.3
Calculated	2.1	2.7	2.8	2.8	2.8	2.6	2.6	2.1	5.3	5.1
E	−0.2	0.1	−0.2	−0.2	0.2	0.7	0.2	−0.2	−0.3	−0.2
Ammonium sulphate, lb. per ton:										
Observed	2.5	5.5	15.4	16.9	19.8	14.3	14.5	8.5	21.0	20.1
Calculated	2.7	6.0	15.7	17.5	17.5	15.4	15.4	9.0	19.2	18.8
E	0.2	0.5	0.3	0.6	−2.3	1.1	0.9	0.5	−1.8	−1.3
1½-in. shatter index, per cent:										
Observed		94.6	96.7	93.9	95.7	77.6	87.3	68.9	79.5	84.1
Calculated		95.8	95.7	92.6	94.6	78.6	88.1	70.4	74.5	76.4
E		1.2	−1.0	−1.3	−1.1	1.0	+0.8	1.5	−5.0	−7.7
¼-in. tumbler index, per cent:										
Observed		61.7	62.9	66.5	63.9	72.6	70.6	75.8	71.8	72.8
Calculated		55.0	65.7	68.6	68.6	75.0	75.0	78.5	72.7	73.0
E		−6.7	2.8	2.1	4.7	2.4	4.4	2.7	0.9	0.2

products obtained from this treatment of coal. The relations given in this paper are almost without exception based on simple linear relations, which it is realized is only an approximation. As more data become available for the lower-rank coals, the equations presented here may be improved.

The data presented in the Bureau of Mines assay tests do not indicate that either coal petrography or blending of coals is important in carbonization except in so far as they are reflected in the proximate analysis. This statement may be qualified by saying that it is based on the facts that the from the average. Furthermore, for the coals for which the petrographic analysis was given, a significant correlation between the deviations from the equations and the petrographic type was found for only a few properties at single isolated temperatures; in these cases the inclusion of the percentage of bright coal as a factor in the equations gave a maximum reduction in the probable error of the calculated values of about 10 per cent.

A final warning should be given in regard to use of the relations given in this report. The numerical coefficients should be re-

stricted to data obtained in identical equipment with the same experimental procedures followed. It may be that some of the relationships given, as, for instance, that of total sulphur in the coke as a function of total sulphur in the parent coal, may be of more general applicability. This remains to be demonstrated. Further, it must be recalled that in the use of the equations the values reported in the proximate analysis for any single coal are interrelated; it is not possible, for example, to regard only the coefficient of moisture and say that for a given coal a change in moisture will produce a certain numerical change in the result, for if the moisture content is changed so also will be the content of fixed carbon, volatile matter, and ash.

It is encouraging to know from the present study that coals are generally so regular in behavior on carbonization that the yields of products and many of the properties of the products can be closely calculated from the results of such a simple test as the proximate analysis. Certainly the proximate analysis is the simplest carbonization assay test.

References

1. A. C. Fieldner and J. D. Davis: U. S. Bur. Mines *Monograph* 5 (1934) 164 pp.
2. A. C. Fieldner, J. D. Davis, R. Thiessen, W. A. Selvig, D. A. Reynolds, F. W. Jung and G. C. Sprunk: U. S. Bur. Mines *Tech. Paper* 570 (1936) 34 pp.
3. A. C. Fieldner, J. D. Davis, R. Thiessen, W. A. Selvig, D. A. Reynolds, G. C. Sprunk and F. W. Jung: U. S. Bur. Mines *Tech. Paper* 571 (1936) 33 pp.
4. A. C. Fieldner, J. D. Davis, R. Thiessen, W. A. Selvig, D. A. Reynolds, G. C. Sprunk and C. R. Holmes: U. S. Bur. Mines *Tech. Paper* 572 (1937) 50 pp.
5. A. C. Fieldner, J. D. Davis, W. A. Selvig, R. Thiessen, D. A. Reynolds, C. R. Holmes and G. C. Sprunk; U. S. Bur. Mines *Bull.* 411 (1938) 162 pp.
6. A. C. Fieldner, J. D. Davis, R. Thiessen, W. A. Selvig, D. A. Reynolds, R. E. Brewer and G. C. Sprunk; U. S. Bur. Mines *Tech. Paper* 584 (1938) 81 pp.
7. A. C. Fieldner, J. D. Davis, R. Thiessen, W. A. Selvig, D. A. Reynolds and C. R. Holmes: U. S. Bur. Mines *Tech. Paper* 595 (1939) 83 pp.
8. J. D. Davis and V. F. Parry: U. S. Bur. Mines *R. I.* 3428 (1939) 32 pp.
9. A. C. Fieldner, J. D. Davis, R. E. Brewer, W. A. Selvig, D. A. Reynolds and G. C. Sprunk; U. S. Bur. Mines *Tech. Paper* 594 (1939) 43 pp.
10. A. C. Fieldner, J. D. Davis, W. A. Selvig, D. A. Reynolds, G. C. Sprunk and H. S. Auvil; U. S. Bur. Mines *Tech. Paper* 596 (1939) 46 pp.
11. A. C. Fieldner, J. D. Davis, D. A. Reynolds, W. A. Selvig, G. C. Sprunk and H. S. Auvil: U. S. Bur. Mines *Tech. Paper* 599 (1939) 38 pp.
12. A. C. Fieldner, J. D. Davis, W. A. Selvig, R. E. Brewer, C. R. Holmes, D. A. Reynolds and G. C. Sprunk; U. S. Bur. Mines *Tech. Paper* 601 (1939) 45 pp.
13. A. C. Fieldner, J. D. Davis, R. Thiessen, E. B Kester and W. A. Selvig: U. S. Bur. Mines *Bull.* 344 (1931) 107 pp.
14. W. E. Deming and R. T. Birge: *Rev. Modern Physics* (1934) **6**, 158–160.
15. F. C. Mills: Statistical Methods. Rev. Ed. New York, 1938. Henry Holt; A. E. Waugh: Elements of Statistical Method. New York, 1938. McGraw-Hill Book Co.
16. V. F. Parry: U. S. Bur. Mines *R.I.* 3482 (1939) 37 pp.; Y. O. Gabinskii and E. S. Krym: *Khim. Tverdogo Topliva* (1936) **7**, 650; C. E. Spooner: *Jnl.* Inst. Fuel (1937) **11**, 134.
17. S. P. Burke, T. E. W. Schumann and V. F. Parry: *Proc.* Amer. Gas Assn. (1930) 820; *Fuel* (1931) **10**, 148.
18. H. H. Lowry: *Jnl.* Amer. Chem. Soc. (1924) **46**, 824; *Jnl. Phys. Chem.* (1929) **33**, 1524.
19. E.g., South Metropolitan Gas Co., Solid Products of the Carbonization of Coal, 123 pp. 1934.
20. H. H. Lowry and G. A. Hulett: *Jnl.* Amer. Chem. Soc. (1920) **42**, 1408.
21. H. E. Blayden, H. L. Riley and A. Taylor: *Jnl.* Amer. Chem. Soc. (1940) **62**, 180.
22. G. Asahara: *Japanese Jnl. Chem.*, (1922) **1**, 35.
23. J. P. Wibaut: *Brennstoff-Chem.* (1922) **3**, 273.
24. T. F. E. Rhead and R. V. Wheeler: *Jnl.* Chem. Soc. (1912) **101**, 831, 846; (1913) **103**, 461; R. F. Strickland-Constable: *Fuel.* (1940) **19**, 89.
25. J. F. Weiler: *Jnl.* Amer. Chem. Soc. (1936) **58**, 1112.
26. H. Eyring: *Jnl. Chem. Physics* (1936) **4**, 283.
27. H. Koppers: *Koppers Mitteilung.* (1932) **14**, 3.
28. J. B. Austin: *Ind. and Eng. Chem.*, Anal. Ed. (1939) **11**, 334.
29. F. Fischer and H. Schrader: *Ztsch. angew. Chem.* (1920) **33**, 172.
30. Methods of the Chemists of the U. S. Steel Corporation for the Sampling and Analysis of Coal, Coke, and Byproducts, 137. Pittsburgh, 1929.
31. A. C. Fieldner, J. D. Davis, D. A. Reynolds, L. D. Schmidt, R. E. Brewer, G. C. Sprunk and C. R. Holmes: U. S. Bur. Mines *Tech. Paper* 604 (1940) 65 pp.

DISCUSSION

(A. J. Boynton presiding)

A. C. Fieldner,* Washington, D. C.—The authors of this paper have rendered a real service to the coal-carbonization and the coal-producing industries in providing a possible means for extending application of the Bureau of Mines survey of the gasmaking, cokemaking and by-product-making properties of American coals to coals for which analyses are available but which have not been subjected to actual tests by the Bureau of Mines-American Gas Association method.

This method was developed for the express purpose of providing comparable data on the properties and yields of gas, coke, and by-products obtained at various carbonization temperatures ranging from those prevailing in

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* Chief, Technologic Branch, U. S. Bureau of Mines. Discussion published by permission of the Director, U. S. Bureau of Mines.

processes of low-temperature carbonization to the customary high temperatures of the gas-manufacturing and coking industries. It was hoped that other investigators would study the results of this survey and extend its usefulness beyond the immediate aims of the Bureau; and it is very gratifying to have Dr. Lowry and his associates make this correlation between the results of the carbonization assay and the proximate analyses of the coals. Other investigators, in turn, will apply the correlation formulas to specific carbonization tests by the B.M.-A.G.A. method in order to determine for themselves the reliability of the calculated results. I have not had any opportunity to make checks myself, but the examples given by the authors in Table 22 show satisfactory agreement for coke, gas, and tar plus light-oil yields. As would be expected, the yields of tar and light oil show somewhat larger deviations than those of the coke and gas. These may be due either to variations in the heating schedule, resulting in more or less cracking of tar, or to differences in constitution of the coal. A greater proportion of spore exines in one coal than in another having the same percentage of fixed carbon would yield a greater percentage of tar and light oil.

Agreement of the observed and calculated 1½-in. shatter indexes for the Pocahontas No. 4 bed coal is surprisingly good in view of the difficulty in closely duplicating individual shatter tests of the same lot of coke. However, the low computed results of the two blends indicate a possibility that the effect of blending may be greater than would be expected by the change in the percentage of fixed carbon of the blend. I have called attention to these differences because I am not ready to accept the conclusion "that the Bureau of Mines assay tests do not indicate that either coal petrography or blending of coals has importance in carbonization except in so far as they are reflected in the proximate analysis." It is admitted that for typical coking coals the percentage of fixed carbon is the most important single factor in evaluating coking properties, but it is believed that the constitution of the coal substance, as conditioned by the nature of the original plant material and its degree of metamorphism, has some influence on coking properties in addition to that indicated by the proximate analysis. Perhaps some coal geologist or paleobotanist interested in this point of view will make a critical study of computed and observed results in relation to the petrographic structure of the coals. The opportunity for an interesting argument seems too good to pass by unnoticed.

Careful attention should be given to the authors' warning that the numerical coefficients used in the formulas should be restricted to data obtained in identical equipment and by the same experimental procedure as was used in the B.M.-A.G.A. method. Further work may demonstrate a wider application of the relations indicated by the proximate analysis. It is generally agreed that no standard laboratory method of carbonization, even on a large unit, can yield results that exactly duplicate those obtained in coke ovens or gas retorts. The commercial results vary with the type of oven or retort. Allowance must be made for such differences in interpreting the B.M.-A.G.A. test results in terms of commercial plants. A comparison of B.M.-A.G.A. tests and commercial-plant yields on 11 coals gives the following results:

1. Plant yields of coke, gas, and B.t.u. of gas per pound of coal usually fall between the test results obtained at carbonizing temperatures of 900° and 1000°C. (1652° and 1832°F.).

2. The amount of light oil scrubbed from the gas of the test apparatus is less than that scrubbed from the gas of commercial plants, because the test condensing train throws down more of the light oil with the tar; however, the total yields at 900°C. from gas plus light oil in the tar are approximately the same as the total from plants.

3. The tar yield from the 1000°C. carbonization shows the best agreement with plant yields, although on several coals the plant yields were 1 to 2 gal. per ton of coal less than those in the test apparatus.

4. The yields of tar and gas from the 18-in. retort usually are closer to those obtained in industrial practice than yields from the 13-in. retort.

5. The ammonia yield from the test apparatus is low because of the decomposition induced by the iron of the retort. The yield of ammonia at 800° approaches that obtained in industrial practice.

6. At a carbonizing temperature of 900°C. the 13-in. retort indicates the relative shatter

and tumbler indexes of the coke, but the figures are lower than those obtained for the same coals in by-product ovens. Much of this difference is eliminated by the use of retorts 18 in. in diameter and a 1½-in. shatter index. The larger retort gives larger pieces of coke that are less fractured, owing to the slower rate of heating.

7. The coke from the test retorts has a lower apparent density and a higher percentage of cells than by-product-oven cokes made from the same coals. Tamping the charge of coal in the retort increases the apparent density and lowers the porosity to figures closely approaching those obtained in by-product plants.

This comparison may be helpful to those who wish to use either the B.M.-A.G.A. observed or computed results in appraising the coking properties of coals.

H. C. Porter,* Philadelphia, Pa.—It appears to be the purpose of the authors to show that different coals, of various types and ranks, are sufficiently characterized by proximate analyses (moisture, volatile matter, fixed carbon and ash) to permit derivation of their behavior in carbonization by formulas based on such analyses. Such a method of assaying the carbonizing possibilities of a coal, if found to have a reasonable factor of correlation with the results of actual practice, would greatly simplify the task of judging coal values, and would be a boon to coal buyers, now dependent on actual oven tests (or the B.M.-A.G.A. large-scale laboratory test) to show carbonizing behavior.

It may be true, as the authors state, that "all coals belong to a family of natural polymers," but it does not follow that, even though the mixture of polymers in one coal may give it a closely similar proximate analysis to that of the mixture in another coal, these coals by virtue of that fact will behave the same in carbonizing —nor that they will exhibit similar behaviors in other respects, as, for example, in plastic qualities under softening by heat, in absorption of oxygen chemically (and incidentally in effect of this on coking properties), in behavior toward extraction by solvents, in content of oxygen and heat value of the dry, ash-free substance. In fact, it is found in many instances that such similarities of behavior do not follow from similar proximate analyses. Coke-oven operators are familiar with the fact that coals having approximately equal contents of volatile matter and fixed carbon (on dry, ash-free basis) often show quite different behaviors in practical cokemaking, as in size characteristics, porosity and strength of the coke, and particularly in blends of coals required for best results in coke quality (examples, from the authors' list, are: coals 42, Upper Banner, with 66.21 fixed carbon; 44, Powellton with 66.0 fixed carbon; and 46, Eagle, with 65.66 fixed carbon).

Such behaviors in coking no doubt are intimately associated with differences in plastic qualities of the coals under softening by heat. It is unfortunate that the authors did not include in their studies the possibility of dependence of plastic properties on simple analyses of the coals, especially proximate analysis as used for the other behaviors in carbonization. This plastic property is a very significant one in cokemaking behavior.

Coking coals, of equal proximate analysis, often also differ in their oxygen content and in the manner in which the oxygen distributes itself (as between hydrogen and carbon) during the breakdown in carbonization. This, as well as plasticity differences, indicates strongly a difference in organic nature of the constituents not shown by proximate analysis.

Calculation by the authors' method of some carbonization yields, and especially of the sulphur to be found in the coke, apparently works out well, with correlation factors of 0.96 to 0.988, but on the industrially important item of gas-B.t.u. yield, although the correlation factor, on the average of tests at 900° and above was 0.91, there were large maximum errors in some cases (for example, coal 10, Illinois, with a calculated yield of 2720 B.t.u., as average of 900° and 1000° tests, against an actual yield in the tests of 3040 and a plant yield of 3140).

Such individual maximum errors are large, even though accompanied, in the whole series, by a good average correlation factor. It is unfortunate that the authors do not give for each one of the test series its own average error. They do state that the average error for all determinations is greater than the probable error by 18 to 20 per cent, and that 50 per cent of the calculated values have a deviation greater than the probable error.

In calculating results to be obtained on physical qualities of the cokes, the authors' method gives large deviations and a poor cor-

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relation factor, as would be expected, since these qualities, like the gas-B.t.u. yields, are greatly dependent on the organic nature of the materials present, and this is not correlated closely with proximate analyses. The average correlation coefficient, in physical coke quality calculations (on 900° cokes) is found to be only 0.62, and there are maximum deviations reaching 5 to 7 times the probable error, and amounting to 1/10 to 1/4 of the magnitude of the determined value.

Such results are in themselves the best foundation for a judgment as to the usefulness of this method for application to the all-important aspects of carbonizing behavior manifested in physical qualities of the cokes and yields of B.t.u. in gas. It appears to be of very limited usefulness, as far as any formulas thus far developed are concerned.

H. H. Lowry (author's reply).—The discussions by Dr. Fieldner and Dr. Porter are much appreciated, especially since they indicate that certain points in the paper have not been emphasized sufficiently.

Dr. Fieldner has called attention to the fact that the agreement of the observed and calculated 1½-in. shatter indexes for the Pocahontas No. 4 bed coal (Table 22) is "surprisingly good in view of the difficulty in duplicating individual shatter tests on the same lot of coke" and has stated that the low computed results of the two blends "indicates a possibility that the effect of blending may be greater than would be expected by the change in the percentage fixed carbon of the blend." These statements require a clear recognition of the fact that the original publications of the Bureau of Mines do not give any numerical values of the reproducibility of the test data and that, therefore, it is still uncertain whether "the low computed results of the two blends" are outside the limits of experimental error of the assay test itself. The point here in question is not one of duplication of individual shatter tests on the *same lot of coke*, but rather one of duplicability of the resistance to shatter of cokes made at different times from the same lot of coal. The data obtained in the tumbler test, which is also a measure of physical quality of coke, show that the errors of the calculated values for the blends are numerically less than the error of the calculated value for the straight coal. Further, Eqs. 20, 21 and 22, which are those derived for calculation of shatter and tumbler-test data, include terms in both moisture and ash, as well as fixed carbon, so that one should not expect to be able to estimate the effect of blending from the change in the percentage of fixed carbon alone.

That coal petrography does not in itself appear from the analysis of the data reported in the paper to have importance except as reflected in the proximate analysis is not unexpected in view of the fact that, while in general the petrographic components of a coal from a given seam may differ in chemical composition, the differences are not quantitatively or qualitatively constant when coals from various seams are compared. Dr. Fieldner has stated that "a greater proportion of spore exines in one coal than in another having the same percentage of fixed carbon would yield a greater percentage of tar and light oil." Again, attention should be called to the equation derived for calculation of the yield of tar plus light oil (Eq. 23), which includes terms in both volatile-matter content and moisture content; one should not attempt, therefore, to calculate yield of tar plus light oil from the fixed carbon. A recent publication of the Bureau of Mines[32] includes both the analysis and the yield of "tar and oil" obtained by the Fischer low-temperature carbonization assay of spores separated from a coal from Williamston, Mich. From the analysis given and Eq. 23, using the constants for 500°C., the calculated yield of tar and light oil in the B.M.-A.G.A. test would be 24.65 per cent. From the data given in *Monograph* 5 (ref. 1) a conversion factor of 0.68 ± 0.04 should be used for calculating the yield of tar plus oil in the B.M.-A.G.A. test from that in the Fischer assay. Use of this factor gives a calculated Fischer assay yield of tar plus oil from the spores of 36.3 ± 2.2 per cent compared with the observed yield of 37.8 per cent. This calculation indicates that normal variation of spore content of coals would not limit the applicability of Eq. 23 for calculation of yield of tar plus light oil from the proximate analysis without reference to the spore content of the coal.

Dr. Porter has pointed out that "coals having approximately equal contents of volatile

[32] G. C. Sprunk, W. H. Ode, W. A. Selvig and H. J. O'Donnell: U. S. Bur. Mines *Tech. Paper* 615 (1940). 59 pp.

matter and fixed carbon (on dry, ash-free basis) often show quite different behavior in practical cokemaking, as in size characteristics, porosity and strength of coke, and particularly in blends of coals for best results in coke quality." The analysis of the B.M.-A.G.A. data given in the paper is in entire agreement with this statement, in view of the fact that the correlating equations developed for calculating coke qualities from analysis include terms in ash and moisture. Coal as carbonized is not dry, ash-free, therefore it is only logical that fixed carbon, or volatile matter, on this basis should be an inadequate criterion of the nature of the products obtained from carbonization.

In criticizing the possible usefulness of the equation developed for calculation of the "industrially important item of gas-B.t.u. yield," it is unfortunate that Dr. Porter chose as an example coal 10 from the Illinois No. 6 seam, for this is one of the lowest rank coals included in the survey and is not used in normal coke-oven operation. It is explicitly stated in the paper that the relations derived are most uncertain for lower-rank coals, and it is pointed out that more data are needed to develop correlation equations to include the lower-rank coals.

While this is not the place to discuss the general theory of errors, Dr. Porter's emphasis on the magnitude of the maximum errors may lose significance on consideration that every step in the B.M.-A.G.A. assay test is subject to an experimental error of a magnitude at present unknown. The analytical errors of the sample probably are not of great significance in this case, though it is entirely undetermined how representative the laboratory sample for analysis is of the individual charge to the retort. The magnitude of the errors in the procedure is not known, as pointed out in the paper, but may be very large in comparison with the basic assumptions mentioned in the paper. Errors are associated also with the actual determinations of yields and properties of the products. Since errors may be cumulative as well as compensative, it should not be surprising to find large errors occasionally. The possible usefulness of the correlating equations should be judged, therefore, rather by the magnitude of the probable errors of the calculated values than by the magnitude of the maximum errors.

In reply to Dr. Porter's last comment, the usefulness of the method described in the paper for calculating the carbonizing behavior of coals when subjected to relatively closely controlled procedures, such as the B.M.-A.G.A. assay, can best be judged by a critical examination of the paper itself. To my knowledge, the method has not yet been applied to data obtained in commercial operations, therefore the usefulness of the method for commercial practice is yet to be determined. In this regard there is a fundamental difficulty, which must first be overcome. The analysis reported in the paper emphasizes that the carbonizing temperature—that is, the final maximum temperature reached by the coke—is the factor that determines the yields and properties of the products from a given coal. In practice this temperature is not generally directly either controlled or measured. The temperature that is controlled is the flue temperature and in many existing ovens this varies over a wide range from one end of the oven to the other and from top to bottom of the oven. Any given oven charge is thus subjected to a range of carbonizing conditions, and this fact alone may well obscure the correlation of the results obtained with analysis of the coal charged. Until this condition in practice is remedied, the usefulness of the method described may well be limited.

Bituminous Coal Production at Varying Levels of Business and Its Relative Use Value as Compared with Former Years

By D. P. Morton,* Member A.I.M.E.

(New York Meeting, February 1941)

Since 1923, which closed the speculative era in the bituminous coal fields of the United States, there have been wide annual fluctuations in the national production of bituminous coal. These changes in annual output have been variously attributed to: (1) increased competition from petroleum products, natural gas and hydroelectric power; (2) lower unit consumption of coal; and (3) fluctuations in the general business level. The purpose of this paper is to measure, from a practical standpoint, the net effect upon the annual trend of bituminous coal production, and to give those who are interested a basis for estimating in advance the probable output of bituminous coal.

It can readily be shown that changes in the level of general business affect the national production of bituminous coal more than does any other factor. Therefore it is necessary to use some standard measure of the relative business level for purposes of comparison. The Federal Reserve Board Index of Industrial Production offers the basis for such a measure.[1] This Index, a weighted composite of 81 different series, rests upon the averages of the years 1935 to 1939 as 100, and is presented graphically in Fig. 1.

Even a cursory examination of the chart will indicate a definite uptrend for industrial production over a period of years. This uptrend is due to at least two factors: (1) an actual increase in national population, and (2) a probable increase in the per capita consumption of goods. This means that there is no such thing as a fixed Normal, rather Normal is a variable, tending to increase from year to year. In order to measure the amount of annual increase in normal, the Index of Industrial Production must be studied in relation to its major components, Durable Goods and Nondurable Goods. Durable Goods are those made from the lasting materials such as iron, steel, other metals, lumber, stone, cement, clay, glass. Nondurable Goods are those for early consumption, such as foods, textiles, tobacco, chemicals, paper, gasoline, and goods made of leather and rubber. The purchase of Durable Goods may be postponed, but Nondurable Goods must be purchased currently unless the standard of living is lowered.

The Federal Reserve Indexes of durable and nondurable goods are shown graphically, 1920 to 1939, on Fig. 2 (pp. 764–765 of ref. 1). It is apparent that fluctuations in the production of nondurable goods are not nearly so wide as in the production of durable goods, and that the curve of Industrial Production (Fig. 1) would fall between the two curves on Fig. 2. It would, however, seem logical that both curves should follow the same long-term trend, or that over a period of two to three decades the relative proportion of durable and nondurable goods, in periods of *normal* production, should be the same. It is neces-

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[1] References are at the end of the paper.

sary to make this assumption for the present because long-term trend can be calculated only from the nondurable index, there having been no recent period of apparent recovery in durable goods.*

three groupings: primary, secondary, and tertiary peaks, as follows:

(1) Primary: January 1920 to June 1929—113 months, with index rise 69 to 95.

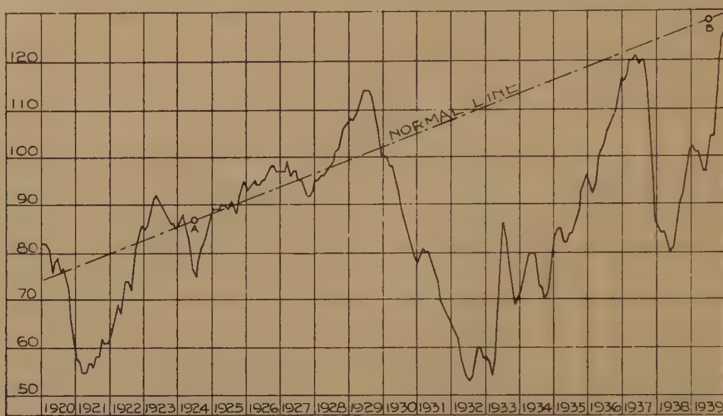

FIG. 1.—FEDERAL RESERVE BOARD INDEX OF INDUSTRIAL PRODUCTION.

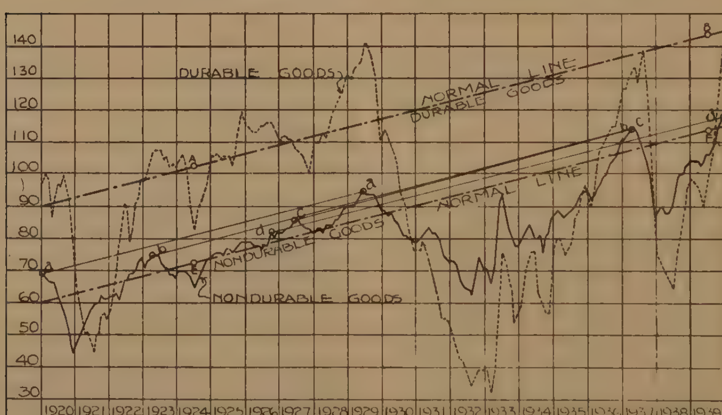

FIG. 2.—FEDERAL RESERVE BOARD INDEXES OF DURABLE AND NONDURABLE GOODS.

The Index figures for Nondurable Goods (pp. 764–765 of ref. 1) in the 20-year period 1920 to 1939 shows the following widely separated peaks: January 1920 = 69; April 1923 = 75; September 1926 = 82; June 1927 = 86; June 1929 = 95; April 1937 = 114; and December 1939 = 118. Plotted to scale, these fall naturally into

(2) Secondary: April 1923 to April 1937—168 months, with index rise 75 to 114.

(3) Secondary: June 1927 to April 1937—118 months, with index rise 86 to 114.

(4) Tertiary: September 1926 to December 1939—159 months, with index rise 82 to 118.

Note that group 1 tests the uptrend for the prosperity period of the twenties; 2, 3 and 4 start from intermediate points in group 1, extend to the two peaks reached in the

* The Index of Durable Goods in September 1940 was 145, the highest on record and apparently still rising.

thirties, and test the current uptrend as well as tying the uptrends of the two decades together. If the lines joining the two points in each pair are essentially parallel, the weighted slope of such lines should be the measure of the long-term trend for the production of nondurable goods.

By the formula $Y - Y_1 = m(X - X_1)$, the slopes of these lines are given in Table 1.

TABLE 1.—*Slopes of Lines*

Slopes	Weights, Per Cent	Weighted Slope
(1) $69 - 95 = m(0 - 113)$ or $m = 0.230$	20.25	0.0466
(2) $75 - 114 = m(0 - 168)$ or $m = 0.232$	30.10	0.0698
(3) $86 - 114 = m(0 - 118)$ or $m = 0.237$	21.15	0.0502
(4) $82 - 118 = m(0 - 159)$ or $m = 0.226$	28.50	0.0644
Total.........558 months	100.00	0.2310

Lines 1 and 2 vary only 1 per cent from parallelism, while lines 3 and 4 vary only 5 per cent. The means of lines 1 and 2 and of 3 and 4 vary only 0.2 per cent from parallelism. Therefore a slope of 0.23 per month, or 2.76 per year, should be a practical value for measuring the uptrend over the past 20 years and for the years immediately ahead. This means that each month's seasonally adjusted Normal for Nondurable Goods should be 0.23 points above the preceding month, and each year's Normal should be 2.76 points above the preceding year.

It is assumed that this same slope will apply ultimately to the curve for Durable Goods, and since the sum of durable and nondurable goods carry a weight of approximately 85 per cent (p. 761 of ref. 1) in the Federal Reserve Board Index of Industrial Production, this same trend should be applicable to it. We therefore have this formula for figuring Normals at any time on either of the three indexes:

$$Y - Y_1 = 0.23(X - X_1)$$

where Y = normal at the basing point,
Y_1 = normal for the desired month,
X = 0.5 or the basing point,*
X_1 = number of months from the basing point.

In selecting the location of the Y axis, or the basic Normal, it is obvious that a wide latitude of choice is possible. Many statisticians agree, however, that the monthly averages of the 3-year period, 1923, 1924, and 1925, represented Normal business conditions for that period. Accepting this as a normal period, the Normal for Industrial Production becomes 86.9 (*A* on Fig. 1); for Durable Goods 102.3 (*A* on Fig. 2); and for Nondurable Goods 72.6 (*E* on Fig. 2) at the mid-point of the period, or June 30, 1924.* Using the formula given above, the Normals for June 1939 become:

Industrial production........................	$86.9 - Y_1 = 0.23(0.5 - 180)$	or	128.2 (*B* on Fig. 1)
Durable goods................................	$102.3 - Y_1 = 0.23(0.5 - 180)$	or	143.6 (*B* on Fig. 2)
Nondurable goods.............................	$72.6 - Y_1 = 0.23(0.5 - 180)$	or	113.9 (*F* on Fig. 2)

The Normal lines arrived at are shown on Figs. 1 and 2.

In studying bituminous coal production, it is desirable to use a Business Index that will express each month's Industrial Production in terms of its percentage relationship to *normal*, rather than to use the Federal Reserve Board Index, which is related to a *fixed* period (1935–1939) obviously subnormal. Such an index can be calculated by dividing the Index of Industrial Production by the computed value of Normal for each month. These values are given by months 1924 to 1938 (excluding 1932) in Column 1 of Table 2.

* Indexes are based upon the mean activity of a month, representing a middle of the month index. A mean activity for three years is ½ month advanced from a median June.

TABLE 2

Column 1—A Business Index Related to Normal = 100.0. Derived from Federal Reserve Board Index of Industrial Production.

Column 2—Actual Bituminous Coal Production (1000 Tons) Needed for U. S. Requirements plus Exports.

Column 3—Bituminous Coal Production (1000 Tons) of 1938 Equivalent Coal, Needed for U. S. Requirements plus Exports.

Column 4—Average Daily Normal Bituminous Coal Production (1000 Tons) of 1938 Equivalent Coal Needed for U. S. Requirements plus Exports.

Month	1924				1925				1926			
	1	2	3	4[a]	1	2	3	4	1	2	3	4
Jan.	100.3	54,707	46,009	1,764	100.6	52,140	45,466	1,738	102.0	57,205	51,027	2,051
Feb.	102.4	49,502	41,631	1,626	100.3	39,270	34,243	1,423	102.8	48,680	43,423	1,760
March	99.9	43,488	36,573	1,408	100.1	39,416	34,371	1,321	103.5	48,244	43,034	1,540
April	96.2	32,629	27,441	1,141	101.0	35,514	30,968	1,226	102.2	43,738	39,014	1,527
May	92.4	34,476	28,994	1,162	100.6	37,276	32,505	1,243	102.0	37,227	33,206	1,252
June	88.7	32,758	27,549	1,242	99.4	35,260	30,747	1,190	102.9	39,201	34,967	1,307
July	86.2	34,646	29,137	1,300	101.3	37,662	32,841	1,247	102.5	38,898	34,697	1,302
Aug.	89.4	37,223	31,305	1,347	100.0	42,933	37,438	1,306	104.5	42,366	37,790	1,391
Sept.	92.6	42,876	36,059	1,558	97.5	44,056	38,417	1,576	105.4	44,658	39,835	1,512
Oct.	93.5	48,914	41,137	1,630	101.6	50,407	43,955	1,602	105.0	46,372	41,364	1,515
Nov.	95.5	42,602	35,828	1,563	103.6	49,997	43,597	1,753	103.7	51,074	45,558	1,757
Dec.	98.7	46,766	39,330	1,533	104.4	52,022	45,363	1,671	103.5	49,141	43,834	1,629
Total	94.7	500,587	420,993	1,439	100.9	515,953	449,911	1,452	103.3	546,804	487,749	1,538
	1927				1928				1929			
Jan.	103.2	49,028	44,468	1,724	98.3	48,925	45,207	1,840	108.5	54,298	50,986	1,807
Feb.	103.0	45,446	41,220	1,667	98.1	43,622	40,307	1,644	108.3	50,037	46,985	1,808
March	104.8	52,747	47,842	1,691	98.8	46,268	42,752	1,603	109.1	41,968	39,408	1,389
April	101.5	38,069	34,529	1,361	98.6	34,910	32,257	1,363	109.8	38,565	36,212	1,319
May	102.2	39,156	35,514	1,337	99.4	39,418	36,422	1,357	111.5	41,908	39,352	1,307
June	102.0	40,383	36,627	1,381	100.1	38,746	35,801	1,376	113.3	39,771	37,345	1,318
July	99.7	36,805	33,382	1,339	101.0	37,064	34,247	1,356	113.0	39,879	37,446	1,275
Aug.	99.4	40,641	36,861	1,373	102.8	41,974	38,784	1,397	112.7	43,195	40,560	1,333
Sept.	98.1	40,863	37,063	1,511	103.5	42,171	38,966	1,569	111.6	43,834	41,160	1,537
Oct.	95.8	44,427	40,295	1,618	105.3	49,876	46,085	1,621	108.4	49,874	46,832	1,600
Nov.	95.5	43,368	39,335	1,648	107.0	47,088	43,509	1,627	103.1	46,314	43,489	1,687
Dec.	96.4	44,014	39,921	1,593	107.8	44,383	41,010	1,522	98.0	46,846	43,988	1,795
Total	100.1	514,947	467,057	1,519	101.7	514,445	475,347	1,522	108.9	536,489	503,763	1,511
	1930				1931				1933			
Jan.	97.8	53,014	50,469	1,985	74.3	41,949	40,649	2,104	52.5	28,568	27,825	2,120
Feb.	97.5	42,360	40,327	1,723	75.1	34,137	33,079	1,835	51.5	30,615	29,819	2,413
March	95.4	38,530	36,681	1,479	76.8	36,626	35,490	1,777	48.6	27,113	26,408	2,012
April	95.2	36,618	34,860	1,465	75.7	31,377	30,404	1,607	52.2	20,905	20,362	1,625
May	93.1	36,713	34,951	1,390	75.5	27,013	26,176	1,333	61.1	23,231	22,627	1,372
June	90.0	34,445	32,792	1,465	73.5	27,891	27,026	1,414	69.8	24,461	23,825	1,313
July	85.9	34,258	32,614	1,460	71.4	29,303	28,395	1,530	76.9	27,775	27,053	1,407
Aug.	83.8	34,717	33,050	1,517	69.4	29,058	28,157	1,560	73.2	28,521	27,780	1,398
Sept.	81.7	37,726	35,915	1,758	65.5	30,455	29,511	1,802	68.5	26,215	25,533	1,491
Oct.	79.6	43,814	41,711	1,941	63.5	34,475	33,406	1,948	64.8	30,894	30,091	1,786
Nov.	77.5	38,409	36,565	1,966	62.5	30,726	29,773	1,985	61.2	30,484	29,691	1,941
Dec.	75.4	40,022	38,101	1,944	61.4	30,879	29,922	1,874	61.9	31,849	31,021	2,005
Total	87.7	470,626	448,036	1,673	70.4	383,889	371,988	1,729	61.9	330,631	322,035	1,734
	1934				1935				1936			
Jan.	63.5	33,359	32,392	1,962	71.5	39,579	38,946	2,095	80.0	44,126	43,332	2,083
Feb.	66.1	34,460	33,461	2,112	73.1	34,949	34,390	1,960	77.3	45,137	44,324	2,294
March	69.5	42,375	41,146	2,193	72.9	32,670	32,147	1,696	78.8	33,238	32,640	1,593
April	70.2	24,061	23,363	1,387	70.3	24,434	24,043	1,368	82.8	32,263	31,682	1,531
May	70.1	26,645	25,872	1,367	70.1	27,655	27,213	1,438	84.4	27,297	26,806	1,222
June	69.1	24,898	24,176	1,346	71.7	24,660	24,265	1,354	85.9	28,944	28,423	1,273
July	63.6	23,951	23,256	1,463	71.5	22,811	22,446	1,207	87.4	31,014	30,456	1,340
Aug.	62.7	26,500	25,732	1,520	74.0	26,722	26,294	1,316	88.1	31,478	30,911	1,349
Sept.	60.8	26,208	25,448	1,744	75.5	24,821	24,424	1,348	89.5	35,187	34,554	1,544
Oct.	61.6	30,308	29,429	1,769	78.7	39,020	38,396	1,807	90.2	41,021	40,283	1,654
Nov.	62.2	30,377	29,496	1,897	79.5	33,847	33,305	1,676	93.3	40,368	39,641	1,770
Dec.	65.5	34,426	33,428	2,041	80.9	38,705	38,086	1,883	95.7	43,115	42,339	1,702
Total	65.4	357,568	347,199	1,732	74.1	369,873	363,955	1,594	86.1	433,188	425,391	1,610

[a] In calculating Average Daily Normals (column 4) Sundays and the following holidays are eliminated. Jan. 1, Apr. 1, July 4, Labor Day, Thanksgiving, and Christmas.

TABLE 2.—(*Continued*)

Month	1937				1938				1939			
	1	2	3	4	1	2	3	4	1	2	3	4
Jan.	95.4	40,938	40,447	1,696	69.2	36,635	36,635	2,118	[b]			
Feb.	96.1	39,461	38,987	1,690	67.5	31,425	31,425	1,940				
March	98.4	45,335	44,790	1,686	67.3	30,380	30,380	1,672				
April	98.2	32,513	32,123	1,308	65.6	23,327	23,327	1,422				
May	98.8	31,971	31,587	1,230	63.9	22,639	22,639	1,363				
June	97.0	33,288	32,889	1,304	64.5	22,598	22,598	1,348				
July	97.7	32,684	32,292	1,271	68.4	23,663	23,663	1,384				
Aug.	97.4	33,687	33,283	1,314	71.5	28,030	28,030	1,452				
Sept.	93.2	37,328	36,880	1,583	72.9	30,869	30,869	1,694				
Oct.	86.5	39,110	38,641	1,718	75.2	33,240	33,240	1,700				
Nov.	76.7	36,399	35,962	1,875	78.9	34,811	34,811	1,765				
Dec.	70.1	38,617	38,154	2,093	79.7	37,328	37,328	1,801				
Total	92.1	441,331	436,035	1,563	70.4	354,945	354,945	1,635				

[b] Business Index for 1939 (column 1) averaged 83.9. Other data for 1939 are not shown as all coal production figures available are preliminary.

Bituminous Coal Production and Theoretical Normal Production

In Table 3 actual United States bituminous coal production is compared with the calculated Business Index from Table 2, by years 1924 to 1939 (excluding 1932), and a bituminous coal Normal Production computed by dividing actual production by the Index. Coal-production figures are taken from U. S. Bureau of Mines or National Bituminous Coal Commission Annual Reports, but production in 1926 is decreased by 20,563,000 tons, and in 1927 by 2,316,000 tons to adjust for estimated surplus exports occasioned by the British Coal Miners' Strike.*

These figures indicate that mean Normal Annual Production of bituminous coal in the first 5-year period was 513,724,000 tons, or 21,757,000 tons higher than that in the last 5-year period. This is an annual drop in Normal Production of 1,978,000 tons. It suggests that there is a downtrend in the use of bituminous coal as a source of energy. Such a conclusion is not justified, since these figures do not take into account the known fact that a ton of coal in 1924 or 1925 had a much lower Use Value than a ton of coal in 1938 or 1939; nor does it account for changes in the amounts of coal held in storage.

TABLE 3.—*Bituminous Coal Production Compared with Business Index*

Year	U. S. Coal Production, 1000 Tons	Business Index	Normal Production, 1000 Tons	Year	U. S. Coal Production, 1000 Tons	Business Index	Normal Production, 1000 Tons	Year	U. S. Coal Production, 1000 Tons	Business Index	Normal Production, 1000 Tons
1924	483,687	94.7	510,759	1929	534,989	108.9	491,264	1935	372,373	74.1	502,517
1925	520,053	100.9	515,414	1930	467,526	87.7	533,097	1936	439,038	86.1	509,957
1926	552,804	103.3	535,142	1931	382,089	70.4	542,757	1937	445,531	92.1	483,758
1927	515,447	100.1	514,932	1933	333,631	61.9	538,981	1938	348,545	70.4	495,108
1928	500,745	101.7	492,373	1934	359,368	65.4	549,510	1939	393,065[a]	83.9	468,494

[a] Preliminary.

Relative Use Values or efficiency factors for bituminous coal can be determined readily for a large portion of annual output (about 45 per cent), but no reliable data are available for the remaining 55 per cent. However, technical improvements in the use of coal are not wholly responsible for the lowered unit consumption that has

* Monthly details given on page 336.

been demonstrated in some categories. A large measure of the savings is accounted for by marked improvement in preparing the coal for market, improvements that were particularly noticeable in the earlier years under consideration. This improvement in market quality would result in fuel savings to all purchasers, regardless of whether or not investment had been made in more modern burning equipment. It seems fair then to assume that the portion of coal consumption for which reliable data are not available has enjoyed at least one-half the weighted economy demonstrated by the portion of consumption for which data are available. In Table 4 there is calculated, on this assumption, a Use Factor for each year, which, when multiplied by the coal produced in that year, will give a use-value tonnage equivalent to that of 1938 tonnage. Basic data are taken from annual reports of the U. S. Bureau of Mines and the National Bituminous Coal Commission.

The Use Factors in Table 4 can be used to adjust actual production figures for a year to the approximate production that would have been required if the coal had been used at 1938 fuel efficiencies, but, to arrive at real relative requirements, further adjustments should be made to correct for fluctuations in the amounts of coal held in storage, and for abnormal production occasioned by the British Miners' strike in 1926 and 1927.

Stored coal, as reported periodically by the U. S. Bureau of Mines or the National Bituminous Coal Commission, can be assumed to have accrued or declined at a uniform monthly rate in periods when monthly data are unavailable. Increases or decreases to stocks are respectively minus or plus changes in actual production, and such adjustments to production figures give real bituminous coal requirements at all times. The figures in column 2 of Table 2 are calculated on this basis, with a further correction for abnormal exports in 1926 and 1927 as follows:

Estimated Abnormal Exports Occasioned by British Miners' Strike

1926	Tons	1927	Tons
June	934,000	Jan.	932,000
July	2,204,000	Feb.	551,000
Aug.	2,591,000	Mch.	464,000
Sept.	2,901,000	Apr.	369,000
Oct.	3,755,000		
Nov.	4,139,000	Total	22,879,000
Dec.	4,039,000		

TABLE 4.—*Use Factor to Convert Bituminous Coal to* 1938 *Equivalent Tons*

Year	Iron and Steel			Class I Railroads			Electric Utilities			Other Uses		Total	
	A	1	B	A	2	B	A	3	B	A	C	A	D
	Million Tons Used	Lb. Coal per Gross Ton Pig Iron	Use Factor of Individual Series	Million Tons Used	Lb. Coal per M Gross Ton-miles	Use Factor of Individual Series	Million Tons Used	Lb. Coal per Kw-hr.	Use Factor of Individual Series	Million Tons Used	½ Col. B Weighted on Col. A	Million Tons Used	Use Factor Col. B and C Weighted on Col. A
1924	65	3,248	0.882	117	149	0.772	38	2.19	0.644	264	0.891	484	0.841
1925	74	3,126	0.917	118	140	0.821	40	2.09	0.675	267	0.913	499	0.872
1926	83	3,048	0.940	123	137	0.839	41	1.95	0.723	286	0.927	533	0.892
1927	74	3,094	0.926	116	131	0.878	42	1.84	0.766	268	0.936	500	0.907
1928	77	3,053	0.938	113	127	0.906	41	1.76	0.801	268	0.949	499	0.924
1929	87	2,984	0.960	114	125	0.920	45	1.68	0.839	274	0.959	520	0.939
1930	70	2,979	0.962	98	121	0.950	43	1.62	0.870	244	0.968	455	0.952
1931	48	2,923	0.980	82	119	0.966	39	1.54	0.916	203	0.979	372	0.969
1932	32	2,933	0.977	67	123	0.935	30	1.51	0.934	178	0.972	307	0.960
1933	40	2,876	0.996	66	121	0.950	31	1.48	0.953	185	0.982	322	0.974
1934	46	2,927	0.979	70	122	0.943	34	1.45	0.972	197	0.980	347	0.971
1935	50	2,838	1.010	71	120	0.958	35	1.45	0.972	204	0.989	360	0.984
1936	66	2,901	0.988	81	119	0.966	42	1.44	0.979	234	0.988	423	0.982
1937	74	2,917	0.982	83	117	0.983	45	1.43	0.986	226	0.992	428	0.988
1938	46	2,865	1.000	70	115	1.000	40	1.41	1.000	185	1.000	341	1.000

The real coal production needed at a given time to satisfy consumption requirements and exports of the United States (column 2 of Table 2) must now be multiplied by the Use Factor for the year in question, in order that comparisons may be made on a relative basis. This calculation will give the tons of coal that would have been required at any time if coal of 1938 Use Value had been used. These 1938 equivalent coal values are given in column 3 of Table 2.

There are, however, still wide variations between annual totals and between totals for the same months in different years. These variations, in large part, are due to fluctuations in the levels of general business (see Business Index, column 1 of Table 2), and to a lesser degree to the variation in the number of available working days in the same month of different years.

In order to eliminate these variables, as far as possible, the monthly values in column 3 of Table 2 may be divided by the Business Index (column 1) and the quotients then divided by the available working days in each month. The net results of these calculations are theoretical Normal daily average bituminous coal productions needed to satisfy United States bituminous coal consumption and exports at any given time. These values are given in column 4 of Table 2. (No effort is made to correct for another variable, abnormal or subnormal temperatures during the heating season.)

In studying these normal daily averages conclusions should be drawn only after careful analysis. It is, though, immediately apparent that these values are higher in years that had a low average business index than in years with a higher average business index. This suggests that the level of general business affects bituminous coal production on a variable scale, and that the effect is on an inverse ratio. It is also apparent that the inverse effect is greater during the heating season (probably because of the relatively constant domestic heating load under average weather conditions) than during the warmer months of the year. Therefore a detailed monthly analysis must be made to be of value. Such an analysis is made by grouping the daily normals (column 4) for each month of the 14 years under definite limits of the business index (column 1), and averaging the groups. The method used is given in Table 5, using July as an example for the entire period.

TABLE 5.—*Mean Daily Normals* (1000 *Tons*) *for July*—1924 *to* 1938—*under Varying Business Indexes*

FROM TABLE 2

Business Index under 70.0			Business Index 70.0 to 79.9			Business Index 80.0 to 89.9			Business Index 90.0 to 99.9			Business Index 100.0 or Over		
Year	B.I.	Normal	Year	B.I.	Normal	Year	B.I.	Normal	Year	B.I.	Normal	Year	B.I.	Normal
1934	63.6	1,463	1931	71.4	1,530	1924	86.2	1,300	1927	99.7	1,339	1925	101.3	1,247
1938	68.4	1,384	1933	76.9	1,407	1930	85.9	1,460	1937	97.7	1,271	1926	102.5	1,302
			1935	71.5	1,207	1936	87.4	1,340				1928	101.0	1,356
												1929	113.0	1,275
Av.	66.0	1,424	Av.	73.3	1,381	Av.	86.5	1,367	Av.	98.7	1,305	Av.	104.5	1,295

In a study of this kind it is impossible to get enough data to be entirely conclusive, and in certain months unusual conditions have lent undue influence to the averages: subnormal temperatures in certain years, and labor disturbances such as the anthracite strike of 1925–1926 and the bituminous

strike of 1935, for example. Elimination of certain months, for any cause, is a matter of judgment, therefore the tabulation in Table 6 is based upon all the data, with no eliminations. The values in Table 6 have been plotted in Fig. 3, and straight trend lines have been fitted to the several points. The use of a straight line may be questioned, since there is some evidence of a flattening of the lines toward horizontal under the high business indexes, but the principal value of these data is to enable the user to predict, within conservative limits, the production of bituminous coal.

The United States is apparently approaching a period when the Business Index may go beyond 110 per cent of Normal. If this develops, and coal production indicates the need for revision of Fig. 3, the lines can be redrawn to meet proven values. In the meantime the lines as drawn offer a conservative basis for estimates.

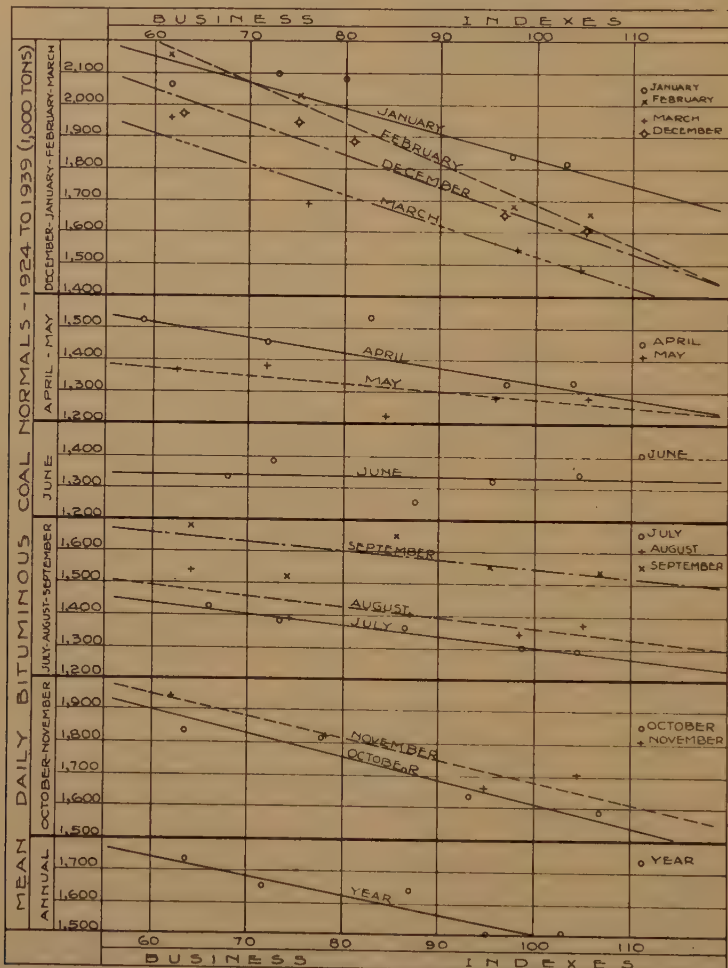

FIG. 3.—RELATIONSHIP DAILY NORMAL PRODUCTION OF BITUMINOUS COAL AND A BUSINESS INDEX, 1924 TO 1938.

The author of this paper has reached the conclusion that there has been a decline,

over the past 15 years, in actual normal demand for total tons of bituminous coal. The drop in demand for tonnage, however, has been caused by the increased use value per ton of coal mined, which has caused the real trend of the effective use of bituminous coal as an energy source to key factors; viz., the probable level of general business, by months, as measured by a relatively reliable business index,* the monthly fluctuations in stockpiles, the surplus exports due at present to war conditions, and the percentage relationship of a future "Use Factor" to the base year

TABLE 6.—*Mean Daily Normals by Months—1924 to 1938—under Varying Business Indexes*

FROM TABLE 2. DAILY NORMALS IN 1000 TONS

Month	Business Index under 70.0		Business Index 70.0 to 79.9		Business Index 80.0 to 89.9		Business Index 90.0 to 99.9		Business Index 100.0 or Over	
	Mean B. I.	Mean Daily Normal	Mean B. I.	Mean Daily Normal	Mean B. I.	Mean Daily Normal	Mean B. I.	Mean Daily Normal	Mean B. I.	Mean Daily Normal
January	61.7	2,067	72.9	2,100	80.0	2,085	97.2	1,840	102.9	1,817
February	61.7	2,155	75.2	2,030			97.2	1,686	105.6	1,657
March	61.8	1,959	76.2	1,689			98.1	1,544	104.4	1,485
April	58.9	1,524	72.1	1,454	82.8	1,531	97.0	1,326	103.8	1,358
May	62.5	1,368	71.9	1,379	84.4	1,222	95.9	1,280	105.4	1,280
June	67.8	1,336	72.6	1,384	87.3	1,258	95.5	1,320	104.6	1,345
July	66.0	1,424	73.3	1,381	86.5	1,367	98.7	1,305	104.5	1,295
August	66.0	1,540	74.3	1,389	87.1	1,404	98.4	1,344	105.0	1,376
September	63.9	1,679	74.2	1,521	85.6	1,651	95.3	1,557	106.8	1,539
October	63.3	1,834	77.8	1,816	86.5	1,718	93.2	1,634	106.8	1,585
November	62.0	1,941	78.2	1,820			94.8	1,660	104.4	1,706
December	62.9	1,973	75.1	1,946	80.9	1,883	96.6	1,656	105.2	1,607
Year	63.6	1,733	71.6	1,633	86.9	1,642	94.9	1,501	103.0	1,508

TABLE 7.—*Bituminous Production Calculated by Formula*

1940	*A*	*B*, 1000 Tons	*C*	*D*	*E*, 1000 Tons	*F*, 1000 Tons	*X*, 1000 Tons	Actual Production, 1000 Tons
January	94.1	1,875	26	1.00	−4,000	+ 250	42,124	44,940[a]
February	89.2	1,825	25	1.00	−1,500	+ 250	39,448	39,105
March	86.0	1,655	26	1.00	−4,000	+ 300	33,306	35,210
April	85.0	1,395	25	1.00	+1,200	+ 500	31,344	32,962
May	87.9	1,305	27	1.00	+2,900	+1,100	34,972	35,468
June	92.4	1,335	25	1.00	+2,400	+ 800	34,039	32,640
July	92.2	1,330	26	1.00	+3,500	+ 400	35,783	36,080
August	92.9	1,385	27	1.00	+3,000	+ 400	38,140	39,240
September	94.9	1,560	24	1.00	+3,000	+ 400	38,931	38,650
Total for 9 months							328,087	334,295

[a] Subnormal temperatures.

remain practically unchanged. It is possible, even probable, that a 2 or 3-year period of near or above normal business conditions may definitely disprove this conclusion. It is, though, doubtful that there will develop any material deviation from the horizontal trend within the next few years.

In using these methods and data it is necessary, of course, to anticipate certain of this study, 1938. Judgment and experience usually indicate business movements,

* If a Business Index, other than the one calculated in column 1 of Table 2, is used, all values in the other tables dependent upon the business index will have to be refigured.

The Cleveland Trust Co., Cleveland, Ohio, publishes an excellent Business Index, which goes back much farther than the one used here. It will be valuable for the study of business cycles, and can be obtained by request.

changes in stocks, or other factors that will lead to reasonably accurate results. When these have been anticipated, it is a simple matter to calculate the theoretical national bituminous coal production for any month by using the following formula:

$$X = \frac{0.01ABC}{D} \pm E + F$$

where X = probable actual U. S. bituminous coal production for a given month.
A = estimated Business Index for the given month.
B = daily normal production from Fig. 3, with A as a base.
C = available working days in the given month.
D = estimated Use Factor for the year in question.
E = additions to or reductions estimated for stored coal.
F = estimated surplus exports.

As an example, this formula can be applied to the first nine months of 1940, since preliminary figures are available for the data to be estimated, and the results listed in Table 7 are obtained. The theoretical production is compared with the actual production as estimated by the National Bituminous Coal Commission.

References

1. Federal Reserve Bulletin (Aug. 1940). Copies can be seen at any Federal Reserve Bank or can be secured from the Federal Reserve Board.
2. U. S. Bur. Mines Am. Repts., Coal in 1924, Coal in 1925, etc., through 1931.
3. Minerals Yearbook 1932–1933—Coal U. S. Bur. Mines.
4. Minerals Yearbooks 1934, 1935, 1936, 1937, 1938, 1939—Coal. U. S. Bur. Mines.
5. National Bituminous Coal Commission, United States Department of Interior.
6. Cleveland Trust Co. Index.

DISCUSSION

(*H. H. Lowry presiding*)

H. G. LANDAU,* Pittsburgh, Pa.—Mr. Morton has shown that production of bituminous coal can be calculated, using mainly an index of the level of general business and a factor for the efficiency with which coal is used. However, there is some question as to the value of this method for purposes of prediction, since it requires that the level of general business be predicted first. Is it not true that to predict "general business" is harder than to predict coal production, because of the much larger numbers of factors that must be considered?

In deriving the relationship shown in Fig. 3, it is not clear why the values for coal production are first divided by the Business Index and then related to this index. Would it not have been just as satisfactory to relate daily coal production (column 3 of Table 2 divided by number of working days) directly to the Business Index for each month of the year? If this relation is a straight line, as it appears to be, it explains the observed flattening of the lines toward horizontal under high business indices in Fig. 3.

* Coal Research Laboratory, Carnegie Institute of Technology.

R. M. WEIDENHAMMER,* New York, N. Y.—Mr. Morton's paper shows such keen analysis of the problem of forecasting the production of bituminous coal that my admiration for his scholarly work leaves but one point of criticism. The chart displayed at this meeting, but not included in the printed paper, indicated a long term downward trend of coal production. This forecast is based on the trend of coal production from 1924 to 1939. Mr. Morton also compares the Federal Reserve Index of durable and nondurable goods from 1920 to 1939, and remarks that there has been no recent period of apparent recovery in durable goods. Today we are witnessing a terrific boom in the output of durable goods, while at the same time the production of nondurable goods may soon be throttled.

Mr. Morton's long-range forecast of a declining curve of coal production appears to be based on too short an experience table. It was during the decade from 1925 to 1935 that the full impact of increased competition from fuel oil and natural gas was felt. Just as the present trend puts an end to the relative downward trend of durable goods, so the burden of proof that coal production should not resume its old growth curve, based on growth of population, even if resumed from a lower level than 1924, still rests with Mr. Morton.

* Cosgrove-Meehan Coal Corporation.

C. E. Mengel,* Bethlehem, Pa.—Among the important factors contributing to the fluctuation in the national production of bituminous is the weather, to which reference is made on page 337, but to which no weight has been given.

From such figures as are available to us, we find that from 20 to 25 per cent of the bituminous production is used for purely domestic heating purposes, and consumption for that purpose no doubt bears some rather definite relation to the variation in temperature during heat-requiring months. We find this is so to a remarkable degree with anthracite (see Fig. 4), and it is definitely so to an even closer extent with fuel oil and gas used for heating.

Table 8.—*Degree Days*

Calendar Year	New Haven	Boston	New York	Philadelphia	Chicago	Chicago Relative to 1931
1931	5180	5191	4630	3877	5172	100
1932	5372	5368	4814	4162	6138	119
1933	5665	5816	5088	4406	5932	114
1934	6018	6347	5531	4873	5973	115
1935	5910	6411	5327	4905	6492	126
1936	5828	6096	5273	4921	6733	130
1937	5502	5740	4996	4709	6724	130
1938	5342	5698	4719	4398	5519	106
1939	5801	6205	5111	4663	5768	111
1940	6315	6529	5758	5312	6754	130

The unit of heat demand that is becoming generally accepted is the degree-day. Table 8 gives this information by calendar years for a number of large cities in areas with which the anthracite business is particularly concerned. This information is available now for virtually every part of the United States, and if the various geographical areas are weighted for distribution, a weighted average demand could be obtained. For example, in Chicago, where the greater part of fuel for heating is, no doubt, bituminous, the index of demand varies from 100 to 130. Assuming, for rough figuring only, that this is typical of the bituminous domestic demand, and that 25 per cent of the bituminous production is used for domestic heating purposes, then the use of bituminous for this purpose may vary from 100 million tons to 130 million tons per year, which is quite a swing.

* Weston Dodson & Co.

There are several minor factors that might be considered; one of them the encroachment of fuel oil. This is, no doubt, more pronounced in the anthracite-consuming area than in the

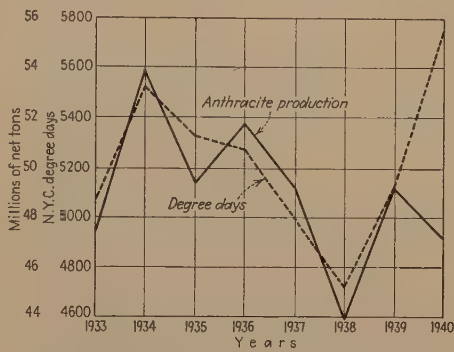

Fig. 4.—Relation of consumption of anthracite for domestic heating to variation in temperature during months burned.

remainder of the United States, but we know that fuel burners are displacing some bituminous-fired furnaces. This has been going on for several years to the extent of possibly 100,000 to 125,000 burners per year in the bituminous area.

The article is a splendid contribution to the economics of bituminous and answers with facts much that has been guesswork as to variation in production. We, in the anthracite business, have been making studies along the same lines; the factors we have to consider are more numerous.

A. W. Thorson,* Detroit, Mich.—The slope of all the lines in Fig. 3 is downward, indicating that as the business index increases the daily normal production of bituminous coal decreases. This seems contrary to my conception of this relationship and I wonder if the author can explain why this occurs.

D. P. Morton (author's reply).—Answering Mr. Landau: There seems to be no question of the interrelationship between the general business level and bituminous-coal production; therefore, when one predicts coal production he is inadvertently predicting business conditions. The business index is a definite value

* Assistant Fuel Service Engineer, Chesapeake and Ohio Railway Co.

at any given time, and the value is readily obtainable. Business movements for the immediate future are reasonably predictable, once a trend is established, and the business level will be the cause while coal production will be the effect. I used the Federal Reserve Board Index of industrial production as a base for my Business Index, in order that readily available data could be used in determining business trends.

Since the use of a constantly increasing Business Normal was the basis of my studies, over a period of years, it was considered advisable to determine daily coal production Normals under varying business levels, so that, with a given business index, the product of the index, days available, and daily normal production would give probable production, assuming no change in use value, stocks, or exports. I now believe that, for general use, the direct relationship of production or consumption to the business index is of more value, and the preparation of a presentation along this line is now under way.

In response to Mr. Weidenhammer: The long-term trend for bituminous-coal production in tons has been downward and recent experience indicates that the trend has not been arrested. However, when we consider the Use Value of that coal, there are indications that the horizontal trend of the 30's has changed to an uptrend. This was pointed out in the presentation. It is also quite possible that, under present conditions, the competitive positions of oil and natural gas may be more favorable to coal, and that the actual production of coal may change to an upward trend. No attempt has been made to project any trend into a distant future, but rather to give the facts as they have affected the recent past.

There is no question of the marked effect temperature changes have upon coal production, as mentioned by Mr. Mengel. The effect is most noticeable upon the domestic demand, but it is also apparent in other classes of consumption. In this paper their effect was disregarded on the theory that the values given represented approximate means. There is now under preparation a study that will attempt to give due weight to temperature variations.

There is also no question of the effect fuel oil has had upon coal consumption, especially on the Eastern Seaboard. In view of tanker shortages, and other factors due to war, it would be confusing to attempt a presentation of this competition now.

In reply to Mr. Thorson: There is a downward slope to all lines in Fig. 3, except for June, which is approximately horizontal. Probably there are two reasons for this:

1. The heating load, over a period of years, is relatively stable, and is not affected materially by business conditions, therefore the proportion of production represented by domestic consumption will be as high as ¼ when production is 325 million tons and business at 60 per cent, and as low as ⅙ when production is 500 million tons and business at 115 per cent. Theoretical normals are determined by dividing production by the index, consequently the low business index has a relatively high normal production, while a high business index has a relatively low normal production.

2. The industrial consumption of coal is probably relatively higher on a unit basis when plants are working at curtailed capacities than when they are working at capacity levels, therefore a plant working at capacity would probably not consume twice as much coal as it would under a 50 per cent capacity basis.

INDEX

(NOTE: In this index the names of authors of papers and discussions and of men referred to are printed in SMALL CAPITALS, and the titles of papers in *italics*.)

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